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**A Practical Treatise,**

EMBRACING

*TECHNICAL AND COMMERCIAL INVESTIGATIONS, THE CHEMISTRY  
AND PHYSICS OF THEORY AND PRACTICE, THE DESIGN  
AND CONSTRUCTION OF EQUIPMENT, AND THE  
OPERATION OF THE PROCESS.*

BY

**H. FORBES JULIAN**

AND

**EDGAR SMART**

M. INST. MINING AND METALLURGY.

ASSOC. M. INST. CIVIL ENGINEERS.

*THIRD EDITION,*

*Revised throughout, Re-set, and greatly Enlarged,*

BY

**A. W. ALLEN,**

METALLURGICAL ENGINEER, MEMBER OF THE INSTITUTION OF MINING AND METALLURGY,  
THE MINING AND METALLURGICAL SOCIETY OF AMERICA, AND THE AMERICAN  
INSTITUTE OF MINING AND METALLURGICAL ENGINEERS;  
AUTHOR OF "MILL AND CYANIDE HANDBOOK."

**With 209 Illustrations.**



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## PREFACE TO THE THIRD EDITION.

AFTER the untimely death of H. Forbes Julian, on the ill-fated *Titanic*, I was invited to collaborate with Edgar Smart in the preparation of a third edition of this most important treatise; and the work was commenced in 1914. A considerable amount of revision was carried out, and a general rearrangement of the matter was agreed upon. The work was then interrupted by the Great War, and the current edition reprinted to meet the continued demand for copies. The joint authorship came to an end in 1916, when Smart was murdered by natives in Africa. Owing to developments in the industry since that date and the need for a greater uniformity in the presentation of the data, a further revision has been made, although some special contributions by both the original authors have been left as presented.

It is fitting that a tribute should be paid to the unselfish and painstaking work of both Julian and Smart; for the original treatise is a monument to successful scientific endeavour. Both authors made it their duty to present facts, without bias or prejudice, disregarding popular theories that were advanced without logical foundation, and taking infinite pains to reach a sound conclusion, even to the extent of carrying out extensive experimental work. The frequent references to "Julian and Smart" as an authority, both in periodical technical literature and in textbooks on the subject, offer sufficient evidence as to the worth of the original treatise. Under these conditions I realise that the revision of the book could only have been carried to an entirely successful conclusion through the instrumentality of the authors themselves.

The third edition will be found to contain much new material dealing with recent modifications in the theory and operation of the process. As previously mentioned, it was found advisable to rearrange the chapters; and the matter, as now presented, is roughly classified into: (1) History, Preliminary Investigations and Theory—Chaps. I. to XX.; (2) Practice—Chaps. XXI. to XXXIII.; (3) Equipment—Chaps. XXXIV. to XLIV. A large amount of material dealing with costs of equipment and operation had been collected, but this has been discarded owing to the wide fluctuation in values since the Great War and the variation in exchange. The data were found to be useless for present-day reference.

The principal additions to the book are in connection with colloidity and adsorption ; the theory of gold precipitation on charcoal ; milling in cyanide solution ; flotation and cyanidation ; zinc-box practice ; the deoxidising of solutions ; counter-current decantation ; aluminium, sodium sulphide, and charcoal precipitation ; agitation, slime-settlement, and filtration equipment ; and a large number of minor references to practically every phase of practice. Additional illustrations and tables have been included, and the Appendix contains a brief account of the use of Martin's antidote for cyanide poisoning.

A. W. A.

LONDON, *April* 1921.

## PREFACE TO FIRST EDITION.

PERHAPS there are no persons who require a work of reference more than those interested in the industry of cyaniding gold and silver ores. The process being comparatively new, a want of data, given in a concise form, has been particularly felt by the practical worker, the investigator, and the engineer.

Although much knowledge on the subject is to be found in various periodicals, it is so scattered and so inaccessible as to be of little service to the busy worker for purposes of reference.

In this book we have tried to bring together a good deal of fragmentary information, which has appeared from time to time in various parts of the world, in books, periodicals, and transactions of learned societies; and which may be of use to the mining engineer, the metallurgist, the chemist, the working cyanider, and the learner. To this we have added information kindly supplied to us by our friends, who have acquired special knowledge in certain departments; and further, we have added many of our own results, obtained from investigations made during the past sixteen years.

Our aim was not merely to provide a formula book of hard and fast rules, nor yet to give only a series of examples of cyanide practice, but rather to lay before the reader some of the principles involved in the working of the process and in the construction of the plant, in order that he may have a better insight into the reasons for certain methods of procedure.

We have endeavoured not to lose sight of the fact that the cyanide process appeals to the investor before the scientist, and such a work as this would, indeed, be very incomplete, if we neglected to deal with the subject from its commercial aspect.

It would not, as is well known, be difficult to obtain more perfect extractions, or to design plant by which material could be handled at a lower cost than is usual, but it often happens that it is not the highest extraction, nor yet the most perfect design of plant, which results in the greatest nett profit in the aggregate. These are points which cannot be too carefully considered, and in trying to impress the reader with this side of the subject, we have dealt with the matter of costs rather more fully than might at first sight appear necessary.

We have attempted to interest the practical worker in fundamental principles, with the object of inducing him to improve and develop the pro-



cess, and to some the discussions may, at first, appear too theoretical for a book of this character. But everyone who tries to improve, acts on a theory of some sort, though not always conscious of it, and on the soundness of this theory success largely depends. A knowledge of theoretical principles is therefore beneficial, if only in conducing to prevent fallacious reasoning.

The manager of a cyanide works is constantly asked to decide on the adoption, or otherwise, of a so-called improvement; and in order to be able to discriminate successfully between the real and the apparent, it is necessary to use sound reasoning, based on theoretical principles, and supported by practical experience. For, to make an alteration without investigating the reason that leads up to it, is little short of gambling; or to adopt a so-called improvement only because it has been successful elsewhere, points to incompetency.

The chemistry of the process being chiefly of a physical nature, we have employed the newer developments in physical chemistry to explain changes that occur in dissolution and precipitation. The application of these modern developments are, we believe, likely to become of far-reaching importance, not only in giving a deeper insight into unsolved problems in the process, but in helping to overcome difficulties that have hitherto appeared insurmountable.

The views put forward regarding dissolution and precipitation are based on experimental evidence. The methods of conducting the experiments and the proofs of the results would be out of place in a book of this sort, and for this reason we simply put forward the views as statements of facts proved only to our own satisfaction. The comparative results given on the dissolution of metals and minerals were obtained from average samples, and are the average of several determinations. This is necessary, because different samples of the same metal or mineral often gave different results; and further, because irregular variations occurred, in the dissolving effect of the solution, at different stages of the investigations; even when extreme care was taken in determining the percentage of cyanide present, and in maintaining the same temperature. The values given should therefore be taken as preliminary, pending further investigations, but for practical purposes they may be regarded as sufficiently correct. In some cases, to save time, the results were obtained by interpolation from empirical formulæ, as this method was found to agree with direct experiment.

We have purposely avoided any abstruse mathematical reasoning, as we are appealing more to those whose knowledge of figures is limited, and there-

fore we have gone but little beyond the rudiments of arithmetic and algebra. Undoubtedly even this extent will be found greater than a certain section of those engaged in the industry command, but we do not think that that is a sufficient reason for omitting what to others may be helpful. And since this book is intended not only for the shiftman, but for the engineer and manager, who, it is presumed, are acquainted with, at least, this slight knowledge of figures, we need say no more.

The formulæ and rules are, for the most part, deduced from practice. Constants are used which may be adjusted to meet abnormal conditions, and thus the formulæ can be universally applied. Copious tables are given, as the best means of simplifying calculations and of expressing results for practical men.

Although great pains have been taken to secure accuracy in the calculations, yet in a book like this, where there are so many figures involved, it is probable that errors have crept in. We should therefore be very grateful to those who would bring to our notice any mistakes.

The illustrations are chiefly from working drawings to scale, reduced, which have been, for the most part, largely used in practice, and sufficient dimensions are given for an engineer to work from.

Much care has been taken in quoting the sources of our information, and we have endeavoured to give full credit where it is due. In some cases, however, a doubt exists as to whom the credit belongs, and then it has been given to the earliest one found published, but in a few cases we were obliged to omit names altogether for want of more definite particulars.

Besides those whose names are mentioned in the text, we are specially indebted to Mr E. P. Martin, B.C.E., New Loch Fyne mine, Victoria, and Mr G. Kermode, B.C.E., Castlemaine, Victoria, for Australian prices; to Mr Charles Butters, A.M.I.C.E., of London, who has placed at our disposal valuable tabulated results of experiments made on a working scale by his staff in America; to Mr Ernest Williams, mining engineer, Johannesburg, Mr John Kelly, consulting engineer, Johannesburg, Mr Clement Dixon, Johannesburg, Mr W. Watkins, A.M.I.C.E., London, Mr M. Torrenti, Johannesburg, and Mr Francis Spencer, Johannesburg, for general technical information.

Our thanks are also due to those secretaries of companies who have shown us much courtesy and given assistance in obtaining costs, etc., amongst whom are Mr J. N. C. Humphreys, secretary to Charles Butters & Co.; Mr Herbert Akers, London, secretary of the Waihi Gold Mining Company; and others whose names are mentioned in the text.

Much as has been done during the past decade in improving the process for treating simple ores and lowering costs, there is still much left to be done before we shall be able to cope with the problem of cyaniding the large bodies of complex ores known to exist. This end will, in our opinion, be most quickly arrived at by constantly bringing together, in a concise form, exact data of researches made under different conditions, and thus facilitating development in the right direction. With this object in view, we earnestly appeal to all connected with the process for short particulars of investigations or working costs which they may wish to submit to us for recording, together with the source of original publication.

In bringing together this fragmentary information concerning the cyaniding of gold and silver ores, we are aware of having fallen far short of our aim, but we trust that this small contribution may be of some use as a guide, and may serve to increase that interest in the process already so strongly manifest in many parts of the world.

H. F. JULIAN.  
EDGAR SMART.

REDHOLME, TORQUAY, ENGLAND.  
P.O. Box 830, JOHANNESBURG, S. AFRICA.  
*December 1903.*

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## PREFACE TO SECOND EDITION.

OWING to the rapid sale of the first impression of this book a reprint had to be made to meet the demand before a revision could be attempted. The exhaustion of the reprint has now given an opportunity for the introduction of a new edition. This contains many corrections and much important new matter.

In order to prevent the book becoming too bulky we have eliminated or condensed portions that have become of less general interest, and have confined the additions chiefly to developments and innovations in the industry. These refer to fine crushing, slime treatment, and details in plant construction. Several working drawings and useful tables have been added, and also a short appendix containing notes on roasting as applied to cyaniding.

Our thanks are due to the numerous correspondents and critics who have kindly brought to our notice obvious errors and misprints.

H. F. J.  
E. S.

*March 1907.*  
*Reprinted 1910.*  
*Reprinted 1913.*

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# CYANIDING GOLD AND SILVER ORES.

## CHAPTER I.

### EARLY HISTORY OF THE CYANIDE PROCESS.

ABOUT the beginning of last century it was first noticed that cyanide of potassium had a solvent action on gold. No practical use seems to have been made of this fact until towards the middle of the century, when Elkington patented his process for electro-plating articles with gold and silver from cyanide solutions. Some interesting investigations were made soon after this by Bragation,<sup>1</sup> Napier,<sup>2</sup> Elsner,<sup>3</sup> and Faraday.<sup>4</sup> These investigations were chiefly of scientific interest, and no suggestion referring to the dissolving of the gold from ores appears to have been recorded until, in 1866, Wurtz<sup>5</sup> mentioned it in a paragraph of a letter. The following year a patent was taken out in the United States by J. H. Rae, entitled an "improved method of treating auriferous and argentiferous ores." The process consisted, in the main, in agitating the crushed ore in a solution of potassium cyanide and simultaneously passing an electric current through the solution to facilitate the dissolving of the gold and silver, and at the same time to precipitate the dissolved metals on cathode surfaces of copper. It seems probable that the use of a cyanide solution to dissolve gold and silver from ores was, to Rae, common property at the time, and that the employment of the electric current was the chief novelty. It is reported that Rae's process was tried on a commercial scale, but with what success is not stated.

Between the granting of Rae's patent and the year 1885 several patents were applied for in which cyanide was used in such a way that the gold and silver in ores were caused to dissolve, but none of these inventions were of importance. During that period a number of investigations had been recorded on the dissolution of gold in ores by means of cyanide. Hahn,<sup>6</sup> in 1870, stated that the gold present in sulphides in ores is soluble in a cyanide solution, and

<sup>1</sup> *Bull. de l'Acad. des Sciences, St Pétersbourg*, 1843.

<sup>2</sup> *Phil. Mag.*, vol. xxv. p. 64, 1844.

<sup>3</sup> *Erdm. Journ. Prak. Chem.*, vol. xxxvii. pp. 441 to 446, 1846.

<sup>4</sup> *Roy. Inst. Proc.*, vol. ii. p. 308, 1857.

<sup>5</sup> *Am. Journ. Science*, vol. xli. p. 222, 1866.

<sup>6</sup> *Chem. Centralblatt*, No. 15.

he pointed out that this fact was taken advantage of in the treating of gold ores. W. Skey,<sup>1</sup> in 1875, made some observations on the action of potassium cyanide on gold ores as then employed in the amalgamation process. He pointed out that the use of potassium cyanide in connection with amalgamation occasioned a loss of gold and silver, as these metals were soluble in the salt. He asserted that both gold and silver in a cyanide solution were electro-positive to mercury, iron, and metallic sulphides found in the ore, although these metals and sulphides were acted on at the same time, but to a less extent. That is equivalent to saying that the gold and silver would be dissolved, whereas the sulphides would be almost unaffected.

It was common practice at that time amongst mill men to treat concentrates for their gold by barrel amalgamation, and it was known that in order to render the gold readily amalgamable the addition of some potassium cyanide had a beneficial effect. Skey observed that the use of cyanide for this purpose was attended by a loss of gold and silver, as these metals were dissolved and were not again precipitated by the mercury or the sulphides, the latter being electro-negative to the gold and silver; but he proposed that in order to avoid loss from this source the fluid used in the barrel should be passed over copper plates. The copper, being electro-positive to gold and silver in cyanide solutions, precipitates these metals.

W. A. Dixon, in a lecture delivered before the Royal Society of New South Wales in 1887, alludes to a patent that had been taken out in America for extracting gold from its ores by dissolving the metal with potassium cyanide, and states that he does not think the process likely to prove satisfactory on account of the cost, the instability and poisonous nature of the salt. These drawbacks had doubtless presented themselves to many minds.

In the year 1885, J. W. Simpson applied for a patent in the United States for a process of separating gold, silver, and copper from their ores, and this consisted in subjecting the ore to a mixed solution of potassium cyanide and carbonate of ammonia, and subsequently precipitating the dissolved metals by means of zinc plates. He points out that his process is particularly applicable to sulphide ores that have been finely pulverised. To remove the precipitated metals from the zinc plates, he suggests agitating them in sulphuric or hydrochloric acid.

It would appear that Simpson added carbonate of ammonia, with the idea that it could be used to facilitate the dissolving of the gold in place of the electric current used by Rae, for he says in his original specification "that by the use of carbonate of ammonia the necessity of electricity is obviated, and the cost of separating the metal from its ores is greatly reduced." He must, however, have found this to be an error, as in the final specification the statement was suppressed. It does not appear that Simpson ever adopted this process on a commercial scale.

About the time that Simpson was applying for his patent, various metallurgists began to interest themselves in the cyanide treatment of gold ores, and

<sup>1</sup> *Trans. and Proceed. New Zealand Inst.*, 1875.

recorded some results. Among these were Endlich and Mühlenberg, W. A. Dixon and L. Janin, jr.

In the year 1887 Wanliiss and Julian, then acting as consulting engineers to the Johannesburg Pioneer Company, erected a small plant to treat blanketings from the company's first stamp mill. It consisted of an amalgamation barrel, a receiver for the treated ore, and an amalgamated copper plate with mercury wells. The barrel was charged with about  $\frac{3}{4}$  ton of blanketings and one-fourth to one-third their weight of water. To this was added about  $\frac{3}{4}$  lb. of 60 per cent. cyanide and 1 lb. of caustic soda. The barrel was then rotated for eight hours, when it was opened and a bottle of mercury (72 lb.), in which  $\frac{1}{2}$  lb. of commercial sodium amalgam had been dissolved, was added. The barrel was again rotated for eight hours and the contents were discharged into a receiver, from which it was slowly washed out with water to collect the particles of amalgam on the plate and in the mercury wells. The gold extracted by this little plant amounted to about 90 oz. per month.

The assays showed that a considerable amount of gold still remained in the tailings undissolved, and it was found impracticable to get anything like a complete precipitation of the dissolved gold in the time allowed. A longer treatment might have been given and a better extraction obtained, but that meant a decreased output of gold, which is a serious item to a small struggling company just beginning operations.

A much more perfect plant was erected at the Roodepoort United Main Reef Company's mill in 1888, which consisted of two barrels, each capable of treating  $1\frac{1}{2}$  ton of blanketings at a time in the same way as in the Pioneer plant, except that the pulp was heated with live steam until it was just too hot for the hand to bear. The ore in this case was less than half as rich as that treated at the Pioneer Company, but the dissolving of the gold was so complete that the washed residue tailing rarely assayed as much as 1 dwt., whereas the original was  $1\frac{1}{4}$  oz. per ton. The difficulty of dissolving the gold was thus surmounted, but the actual gold recovered did not at all tally with the assays, there being a discrepancy, which was found to be due to bad precipitation by the sodium amalgam. About this time Julian was making investigations in connection with the dissolution of gold in ores by chlorine, and also by cyanide under a high pressure of air, and subsequent precipitation by an electric current and by sodium amalgam, which resulted in some patent applications in South Africa. For oxidised ores, the dissolution of the gold by chlorine under pressure of air and precipitation by sodium amalgam was as perfect as could be desired; in fact, some blanketings containing 34 oz. per ton left only about 7 dwt. in the residue, equivalent to a 99 per cent. extraction. The fine gold was dissolved and re-precipitated as amalgam, while the coarse gold was made easily amalgamable. But when pyritic ore was dealt with, the dissolution of the gold became impracticable without an excessive consumption of chlorine, and it was found necessary to again return to cyanide as a solvent; and in order to precipitate the dissolved metals an electric current was employed, the anode being iron or lead

peroxide, and the cathode, mercury. A number of experiments had been carried on at Johannesburg and Kimberley in 1888-89 embodying these latter ideas, dealing with 1-ton lots at a time.

In 1889 a plant was erected by the Wemmer Gold Mining Company, Witwatersrand, to deal, by this process, with ore that contained pyrites. It consisted of three rotating barrels, each having a capacity of two tons of ore. About the same proportion of cyanide and alkali was added as in the previous cases, but a pressure of air was kept up in the barrels of about 30 to 40 lb. per sq. in., and the contents were heated with live steam. The results showed that when the ore was not too coarsely crushed the gold dissolved rapidly and completely within a short time. When practically all the fine gold was dissolved, the pulp was discharged into a circular agitating vat of wood. The bottom of the vat was covered with a well-amalgamated copper plate, coated with an excess of mercury. This formed the cathode of an electrolytic cell. The anode consisted of an iron disc stirrer which kept the ore particles in a state of suspension, and continually brought fresh solution in contact with the mercury cathode. The side of the vat was also lined with iron to within a few inches of the bottom, and connected to form an anode. After the pulp had been agitated and the electric current passed through it for some hours, the vat was slowly discharged through sluice gates arranged at different levels, with the addition of more solution. The pulp then flowed over well-'set' amalgamated plates and through mercury wells or riffles. The baffle boards of the riffles were covered with lead peroxide and connected with the dynamo to form anodes, the mercury and plates being the cathodes. The mercury cathode in the vat soon became sodium amalgam. This was removed at intervals and added to the ore in the barrels to assist precipitation.

It was found that the electrode surface was too small to get anything like complete precipitation in the longest time that could be allowed in practice; and, to compensate partly for this, the solution was circulated and kept continuously coming in contact with the electrodes. The amalgam obtained, either by precipitation with sodium amalgam or by the electric current, was of a fine slimy nature and could not be squeezed, but if allowed to stand in a pail for a few days it assumed, to some extent, the characteristics of ordinary amalgam.

At the time that the small plant erected by Wanliss and Julian was being worked by the Pioneer Company, J. S. MacArthur, R. W. Forrest, and W. Forrest, who had been experimenting in Scotland with samples of gold ore, applied for a patent in Great Britain, No. 14,174, 19th October 1887, for extracting gold and silver from ores by dissolving the metals with a solution containing cyanogen or a cyanide. At a later date they state, in the United States application for patent, that a solution containing from two to eight parts by weight of cyanogen per 1000 of water without the presence of any other chemically active agent "has a selective action such as to dissolve the gold and silver in preference to the base metals." They suggest that the gold and silver may be obtained from the solution by any convenient known

way, "such as evaporating the solution to dryness and fusing the resulting saline residue, or by treating the solution with a sodium amalgam."

It is probable that this patent would never have been noticed by practical men were it not that MacArthur and Forrest followed it up with an ingenious method of precipitating the dissolved metals by zinc. The fact that gold and silver could be dissolved from ores by a cyanide solution did not add anything to the knowledge previously possessed by metallurgists, and the discovery of the so-called 'selective action' of weak solutions is a factor of no practical importance. This is self-evident from the fact that other factors than strength of solution, viz. time and cost of material, have to be taken into account, so that the most suitable strength of solution is often not within the limits of the solutions said to have selective action.

MacArthur and Forrest added, to the gold mining industry, knowledge of great value, which has been highly appreciated by practical men, through the introduction of zinc in the form of fine shavings or turnings for precipitating gold and silver from cyanide solutions. They covered this invention by a patent obtained in Great Britain, No. 10,223, and dated 14th July, 1888, of which the second claim consisted "in precipitating gold and silver from cyanide, chloride, bromide, thiosulphate, sulphate, or other similar solutions, by means of zinc, the employment of the zinc as freshly prepared in a state of fine division." The claim has been considered too broad, inasmuch as in the years 1862-64<sup>1</sup> it was pointed out that zinc in a granular condition is particularly suitable for precipitating from solutions most metals occurring in ores; also, a patent was obtained in Great Britain in 1884 by A. P. Price for precipitating gold and silver from solutions resulting from the treatment of ores by agitating the solution in contact with zinc in a fine state of division. Although these may be considered as anticipations of MacArthur and Forrest's patent, the employment of zinc as used by them in the form of fine shavings was certainly novel, and has much to do with the early success of the cyanide process.

In the year 1888, Werner von Siemens, of Berlin, applied for patents on the chief goldfields of the world for the extraction of gold and silver from ores, which consisted in treating ores with a cyanide solution, and afterwards precipitating the gold and silver from the separated solution by means of an electric current, using anodes of iron and cathodes of lead. The process was not applied on a working scale until 1893.

Thus it would appear that the extraction of gold and silver by the aid of cyanide began from mere suggestions and laboratory experiments, which conveyed to various minds the possibility of devising a process for dissolving precious metals from ores. One of the most obvious factors in favour of cyanide as a solvent, which has been appreciated by chemists from the first, is that the compound formed is of so stable a character that the mineral matter of the ore does not precipitate the dissolved metals, whereas when chlorine is used it is difficult to prevent the gold from re-precipitating. In other words, the gold and silver are more electro-positive in a cyanide solution than the

<sup>1</sup> *Die Chemisch-technischen Mittheilungen*, 1862-63, and Otto's *Chemistry*, 1864.

pyrites or other mineral matter, while in a chlorine solution the reverse is the case.

Had Rae's process of 1867 been applied on a suitable ore and the work followed up by trained metallurgists, there is no doubt but that it would have developed in a few years into a successful cyanide process. As it was, and in spite of additional knowledge and subsequent improvements on Rae's ideas, it was not until the year 1891, when a plant was erected by the Robinson Gold Mining Company to treat tailings by the MacArthur-Forrest process, that the treatment of ores by cyanide had crystallised into a form acceptable to the mining world. From that time it may be said that the cyanide process, which has proved such a phenomenal success, dates.

The chief credit in proving the adaptability of cyanide to gold ores and in putting before the world a workable process is due to MacArthur and the Forrests, their colleagues, and the Cassel Gold Extracting Company of Glasgow; and not a little of the credit is due to that careful chemist, G. A. Darling, who successfully carried out the treatment with the first Robinson plant at a time when the available knowledge of practical difficulties was so meagre that success depended almost entirely on his technical skill. From 1893 onwards the successful application of the cyanide process admits of no parallel in the history of metallurgical advance. In the treatment of gold ores it has superseded the chlorination process and has been mainly instrumental in the opening up of vast avenues of industry in the treatment of ores which, previous to its development, could not be profitably worked by any known method. In the treatment of silver ores it has entirely displaced the *patio* and hyposulphite processes; and in this case, as with the handling of other classes of material, the success met with has been largely due to the application of research to industry. Improvements in method, and scientific results of far-reaching importance, daily help to extend the scope of the process; and the many metallurgists who have undertaken the task of perfecting methods of treatment are to be congratulated, and their work commended, not only on account of the value of the results, but also for the generous and whole-hearted manner in which so many of them have placed the information at the disposal of their confrères throughout the world. To the various scientific societies and technical journals who have helped to disseminate this knowledge the profession is under a debt of gratitude. The introduction of the cyanide process has marked an era in the history of gold and silver mining, not only by diminishing risks to the investor and increasing profits, but by bringing into the industry a large number of professional men who take positions second to none in technology and science.

## CHAPTER II.

### PRELIMINARY INVESTIGATIONS.

#### *Section I.*

IN determining the adaptability of the cyanide process for the treatment of any particular gold or silver ore, it is imperative that the utmost accuracy be observed in all preliminary investigations. There are many important factors to be taken into account, and any one of these may have a special bearing on the construction of the proposed treatment plant, as well as on the method adopted for the reduction of the ore. Amongst the various points which should be considered are the following:--

1. The physical and chemical states in which the gold and silver exist in the ore, as far as determinable.

2. Whether the gold and silver in the ore may be concentrated by gravimetric or other method to an economic extent; and whether the concentrated product, on account of refractory content or high value, should be isolated from the bulk of the ore and treated separately.

3. Whether an economically satisfactory extraction may be obtained by medium or coarse crushing followed by separate treatment of the sand and slime; or whether the fine grinding of the whole of the ore is necessary.

4. Whether roasting of the ore, or any concentrated product, is advisable, and whether this should be done before cyanide treatment.

5. Whether the adoption of a special method of treatment, such as preliminary desulphurising or the addition of bromo-cyanide, is applicable in the instance under review; and whether economies could be effected by any such modification of treatment.

From the results of the metallurgical and physical experiments on the ore, a considerable amount of data should result. These will be of value in outlining a method of treatment and in forecasting the expenses of operation. The ultimate profit is the important consideration, and it is upon this that all decisions affecting operation schemes should be based. To estimate probable cost of plant construction and operation, and the profits which may accrue, with any degree of accuracy from preliminary investigations requires much judgment, and long practical experience in the working of the process.

The several points above mentioned may be subdivided into others; and if each investigation is carefully made and systematically tabulated, it will be found that the trouble and expense it entails will be more than compensated



for, disappointment will be averted, and the success of the selected process practically assured. The importance of such an investigation before proceeding to design a plant presents itself forcibly when economic considerations are taken into account, for in treating a gold or silver ore there is often but a narrow margin between profit, in one case, and loss in the other.

Investigations of this sort should be entrusted only to one who has a thorough theoretical and practical knowledge of the process, skill in laboratory manipulations, and some business ability—in reality, a man who has had a mechanical and chemical training, and practical experience in the complete treatment of gold and silver ores by wet processes.

No hard and fast rules can be laid down as to the method of carrying out an investigation and in treating an ore, but a few general outlines are here given as a guide, and these may be modified to suit any variety of circumstances.

**Sampling.**—The first important matter is to get a portion of the ore to be investigated. This should be taken so as to represent a fair average sample of as many thousand tons as it is practicable to procure. The sample cannot be too carefully taken, for this is the foundation on which the entire investigation rests.

The ore to be sampled may be taken from—

- (a) The mine.
  - (b) An ore heap before crushing.
  - (c) The current product as it leaves the mill in a crushed condition.
  - (d) A heap of material already crushed.
  - (e) Truck loads as they go to or from the works, or from belt conveyors.
- (a) The sampling of the ore in the mine is usually done by breaking pieces off the whole face of each stope at regular intervals. This does not appear to be a difficult task, but considerable judgment and intelligence are required in avoiding an undue quantity of abnormally rich veins or pockets or of more or less barren rock, in order to get a fair sample. Mine sampling is a business in itself, and should be entrusted only to those who are specially qualified for the work and disinterested in the results, but it is well that the metallurgist should know what to do with the sample when it is taken to the surface.

The sample ought to weigh at least 10 cwt., and, if the mine is extensive, may be as many tons. On the surface it is spread out on a hard clean floor, broken into pieces to pass through a 2- or 3-in. ring, and then shovelled into a conical heap. It is next thrown back into another heap, by which time the ore should be properly mixed, and this operation may be again repeated, if thought necessary by the sampler. It is now levelled off into a flat circular-shaped heap about 6 in. deep or more, and two strings are stretched across its diameter at right angles, as in Fig. 1, two opposite quarters are retained as 1 and 2, while 3 and 4 are discarded. The part of the ore which is retained is again turned over and thoroughly mixed, and a similar

circle is made and again quartered down until the sample is finally reduced to 1000-3000 lb. This is then broken by a small hand or power stone-breaker to hazel-nut size, and is again reduced in bulk by quartering, or by means of a Clarkson or Jones divider, to, say, 200 or 400 lb. The sample is crushed to any desired degree of fineness by means of a 'dolly' or hand stamp with water. The whole of the water and slime should be saved. If crushed dry, a small ball mill of the closed type answers the purpose, but care should be taken that as little as possible of the dust escapes. Every piece of ore belonging to the sample should be completely crushed to pass a sieve of the desired mesh, and nothing left.

(b) When an ore heap is to be sampled it is usual to make a cutting right through its centre of a narrow but convenient width to work in; and as the ore is removed, it is dumped into a conical-shaped heap and then dealt with as already described under (a). When this heap is large it is, of course, reduced in bulk by quartering to a convenient size of 2 or 3 tons before breaking down to 2- or 3-in. gauge.

(c) Crushed ore as it leaves the mill may be either wet or dry, the former being most usual with gold ores. When crushed dry, samples may be conveniently taken by an automatic sampler as the ore is discharged from the mill. Such a sampler, to give satisfactory results, is not difficult to construct.

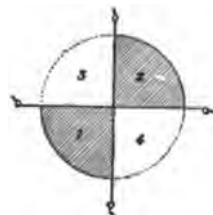


FIG. 1.—Sampling.

When, however, the ore is crushed wet, sampling becomes a more difficult problem. The ore as it leaves the battery tables is usually carried away by a launder, and at any point where a fall may be conveniently obtained, samples are taken. This is done by taking a measured quantity of water and sand at regular intervals of every hour or so, by placing a vessel under the launder so as to take the whole of the pulp then passing, and next discharging the pulp caught into a large tank. This operation may be conducted by hand, but it is easy to devise an automatic appliance for the purpose which answers fairly well. The contents of the large tank are allowed to settle and the clear supernatant liquid is syphoned off at intervals, while the sand and muddy water are run off into a drying pan, where the superfluous water is evaporated and the ore thoroughly mixed. This may be done once a day, or, where the ore is more uniform, once every two or three days.

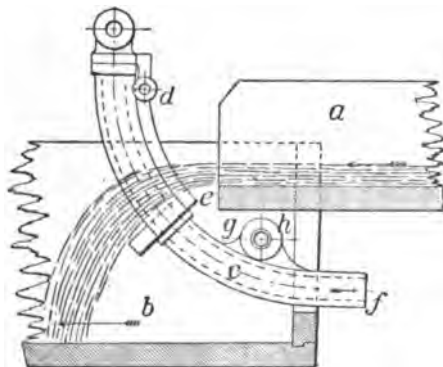
In this way an accurate average may be obtained, but the method requires cumbersome plant, is troublesome to deal with, especially in large mills, and involves much labour after the sample is collected.

In order to obviate these difficulties, a large number of automatic appliances have been devised, whereby small samples are taken continuously or intermittently from a narrow width of the launder, so as to reduce the bulk handled and save labour. An excellent wet sampler has been designed by J. Higham, the essential feature of which is that a vertically slotted pipe is moved horizontally across the stream of pulp at a uniform speed. This

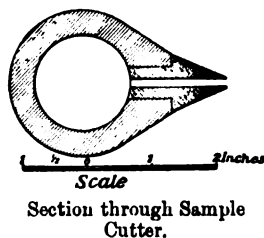
apparatus is illustrated in Figs. 2 and 3, kindly supplied by J. Kelly of Johannesburg.

The pulp from the end of a launder *a* (Fig. 2) delivers into a second launder *b*, thus impinging upon the curved pipe *c*, which is slotted from *d* to *e* on its concave upper side. This pipe *c* is attached to a nut *g* (Fig. 3) threaded on the long screw *h*. Consequently, when this screw is rotated, it imparts a horizontal motion to the sampling pipe across the stream of pulp.

A shaft *i* is driven at a uniform speed by any suitable means, such as an



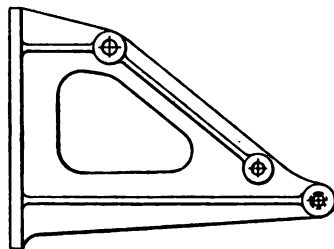
Side Elevation of Sampling Pipe.



Section through Sample Cutter.



Detail of Fence Plate *e*.



Projection of End Standard.

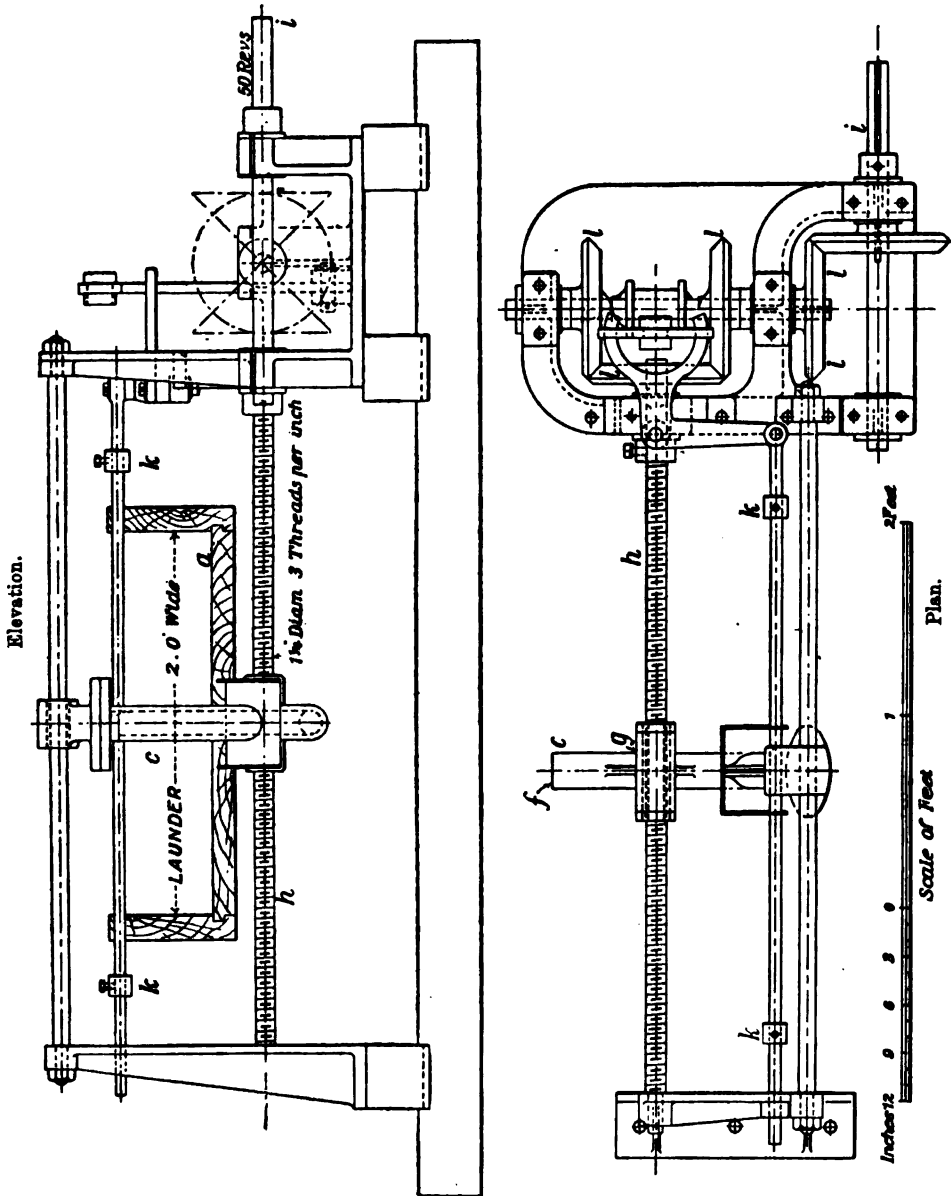
FIG. 2.—Higham's Wet Sampler.

electric motor, and is connected to the screw-shaft *h* through a train of mitre wheels *l l*, provided with a reversing gear automatically operated by the adjustable stops *k k*.

By this mechanism the sampling pipe is caused to travel to and fro through the entire width of the stream, while the slot continuously intercepts a thin slice of the pulp which is delivered at the lower end *f* of the pipe into any suitable receptacle. A fence plate *e* prevents any of the rejected pulp trickling down the outside of the pipe *c*.

The percentage of the whole pulp which is intercepted depends upon the width of the slot, relatively to that of the launder *a*. In the example illustrated the slot is  $\frac{1}{8}$  in. wide and the launder 2 ft., so that the sample is equal to 0.5 per cent. of the material delivered. When this quantity is incon-

veniently large, it may be passed over a second sampler of the same kind, or it may be reduced by using the Higham dividing box shown in Fig. 4.



**FIG. 3.—Higham's Wet Sampler.**

This device is a closed metal trough *m* with a flat inclined top *n* pierced with a number of equidistant slots *o o*, each of which is provided with a pair of knives *p p*, fixed  $\frac{1}{8}$  in. apart and of the form shown in detail in Fig. 4. The dividing box is placed under the delivery end *f* of the sampling pipe *c* in the

line of travel of the latter, so that each slot receives an equal portion of the sample at equal intervals of time, while all which falls on the top of the box between the slots is rejected. In the size shown, the slots are placed at  $2\frac{1}{2}$ -in. centres, so that the reduced sample delivered by the pipe *q* has only  $\frac{1}{20}$  of the bulk of the original sample.

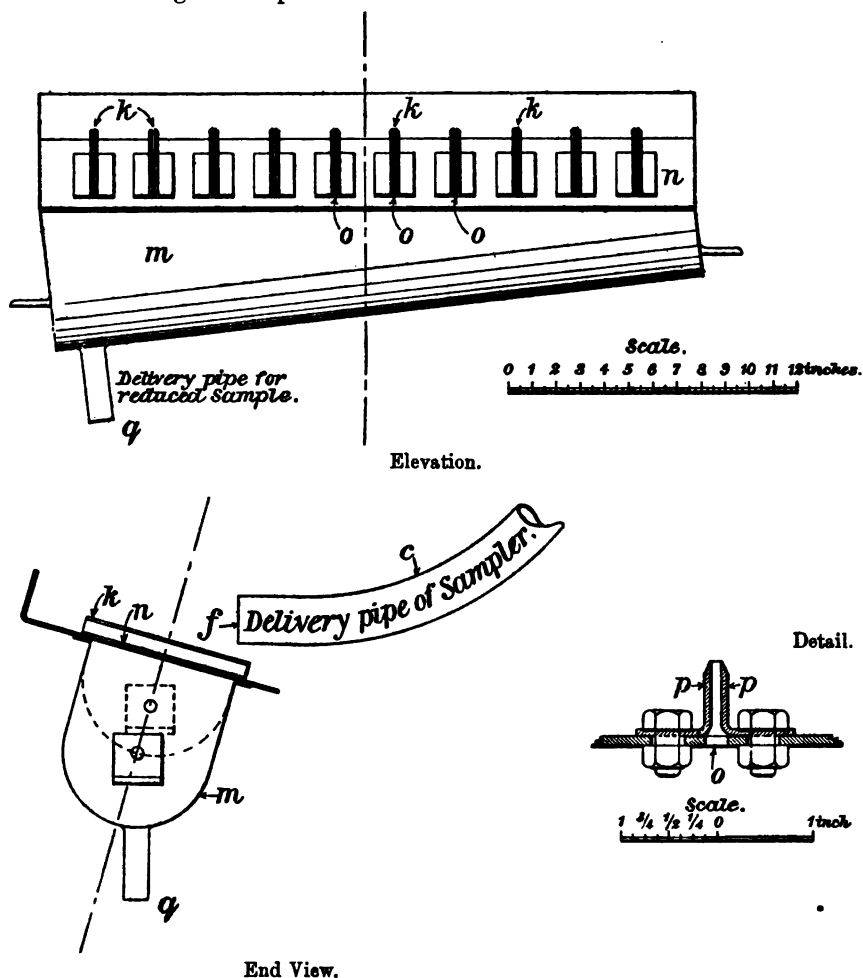


FIG. 4.—Higham's Dividing Box.

The sample should be dried at a temperature not exceeding  $150^{\circ}$  F. until it contains about 5 or 6 per cent. moisture, in which condition it is reduced in bulk to about 200 to 400 lb. by quartering and dividing as described under (a), or better by employing an automatic apparatus such as a Clarkson divider. This apparatus is shown in Fig. 6. It mixes the sample thoroughly, and gives a uniform product of any desired bulk. If the time of collecting the sample be noted, and the ore after drying be weighed,

a fair approximation of the quantity of material being daily produced can be obtained.

A useful instrument for taking accurate hand samples of conveniently small size from slime pulp flowing from the edge of a launder is illustrated in Fig. 5. It consists of a wedge-shaped box *a*, as shown in section, about

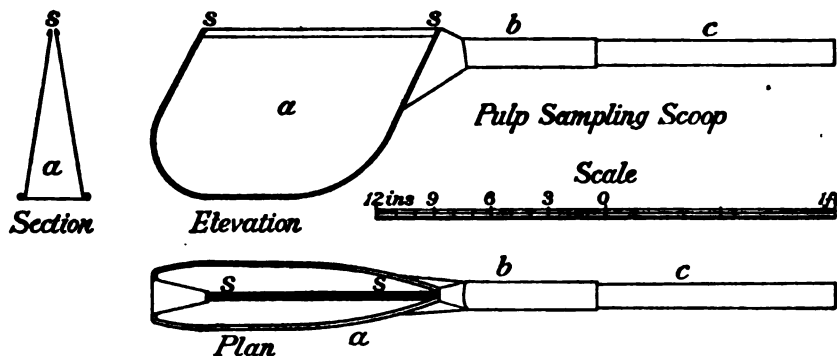


FIG. 5.

12 in. long, 3 in. wide at the back, and 9 in. deep. Along the front edge a slot *s*,  $\frac{1}{8}$  in. wide, is left between the sides. The box is conveniently made of  $\frac{1}{16}$ -in. sheet steel, and is provided with a tabular socket *b* into which a wooden handle *c* is fixed.

The sample is taken by holding the scoop so that the slot faces the launder, and by passing it quickly and evenly across the flow of pulp until full. The contents of the scoop are then immediately emptied into the sample bucket.

(*d*) It sometimes happens that the ore to be sampled consists of a heap of tailings which has accumulated from time to time. In this case the operator proceeds by laying off two sets of equidistant parallel planes 8 to 60 ft. apart, at right angles to each other, thus dividing the whole of the heap into squares of equal area, measured horizontally, as shown in Fig. 7. Pegs are inserted at the corners of each square, and a sampling rod is pushed down in these spots through the material to the surface of the ground beneath it. The whole of the cores drawn from the four holes of each alternate square are well mixed, reduced in bulk if necessary, then put into clean linen bags, numbered, sealed, and sent to the assayer, each set of holes being completed before proceeding with the next.

In this way a fair average of the metal content of the whole heap may be determined, but before this can be done with any accuracy it is necessary to know the cubic contents of the material in each square.

Referring again to Fig. 7, the levels of the surface of the heap and of the



FIG. 6.—Sampling and Dividing Machine.

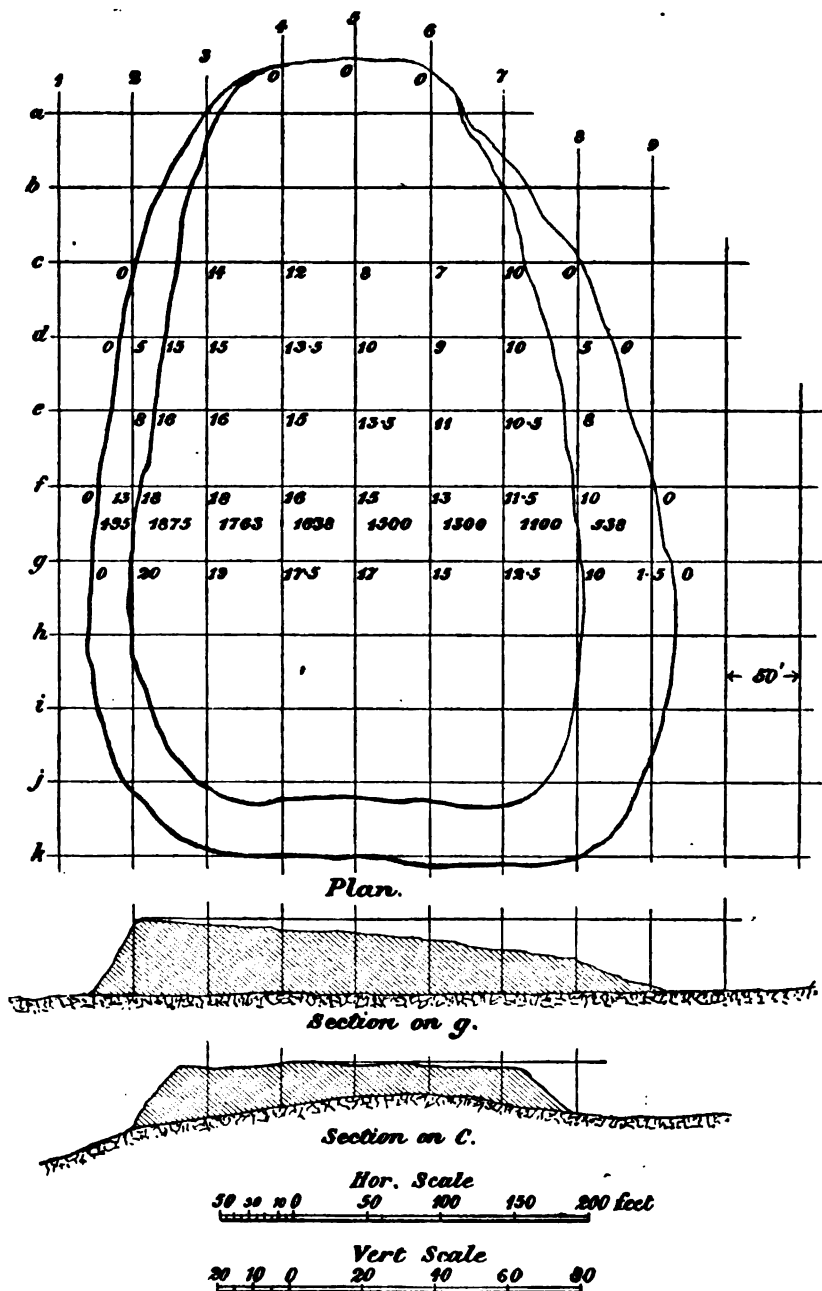


FIG. 7.—Measurement of Heaps.

ground on each side are taken with a level and staff in the ordinary way, and sections plotted showing the actual outline of the tailings and the real shape of the ground along any line of borings. The surface of the ground underneath the heap can often be sketched with sufficient accuracy when the general surface of the country shows the inclinations to be constant, as in the upper sectional view (Fig. 7). If convexity or concavity exist, as in the lower view, this must be determined by borings. The depth of material at the corners of each square, whether determined by direct borings or scaled off the plotted sections, should be marked on the plan at the corner of its respective square, as partly shown in the plan on Fig. 7, and the contents below each square will evidently be the average of the depths at the four corners multiplied by the area, or, what is the same thing, the sum of the four depths multiplied by a quarter of the area. Thus, if the sides of the square were 20 ft., its contents in feet would be the  $\frac{\text{sum of 4 depths in ft.} \times 20 \times 20}{4}$  or

sum of 4 depths  $\times 100$ , and this result divided by the cubic feet in a ton of tailing gives the contents of each square in tons, which should be set down on the plan as shown between lines *f* and *g*. The quantity of gold and silver in the whole heap may then be calculated, and this divided by the total number of tons gives the average content per ton.

It sometimes happens that the heap is too deep for the sampling rod to reach the bottom, in which case several pits are dug at equal intervals until the rod can be driven to the floor. The core drawn from below these pits may be mixed to form separate samples, and assayed and tested separately.

Heaps of concentrates are measured and tested in the same way, but the borings are made much closer, 4 to 8 ft. apart being sufficient.

Often a tank of ore is sampled in the same way, that is, by driving the sampling rod down to the filter cloth at equidistant points, the cores from which are thoroughly mixed and sent to the assayer, and the same method is again repeated on the residues after draining.

This method of taking samples from a tank is objected to by some on account of channels being formed for the solution, instead of the latter flowing uniformly through the whole mass. This objection does not always hold; but there are certainly cases where the method has proved prejudicial, particularly when the ore has not been sized, or contains much slime with coarse sand.

(e) When the ore is conveyed to the tanks by the truck or car load, or when it is carried by a belt conveyor, an excellent opportunity is given to take a fair average sample. In the case of truck loads, a short sampling rod, about 1 in. diameter and 2 ft. long, is driven into each load as delivered, and the cores are collected together. The same method is employed with the residues when discharging into trucks. This method, however, entails considerable expense in labour. When a belt conveyor is used, samples may be taken at regular intervals by filling a small dish and discharging into a large vessel. This may be done automatically.



Discrepancies occur, whatever method of sampling is employed. These are to a certain extent unavoidable, owing to the great number of factors that are involved, many of which may vary unobserved. It is only on rare occasions that the estimated gold and silver in a quantity of ore corresponds with that accounted for after treatment; and when it does so, it is just as likely as not to be a coincidence rather than a sign of careful work. One can, however, nearly always rely on getting a fair approximation when the operations are conducted with care by qualified men, and more particularly where the ore has been previously sized.

**Tailing-Dump Sampling Rod.**—This is usually made from a length of ordinary gas-piping, about  $1\frac{1}{2}$  in. diameter, by cutting a slot lengthwise, taking out rather over one-third of the circumference to within a foot of the top end. A cross-bar about 2 ft. long is then firmly secured to the uncut end, for the purpose of rotating and lifting the rod. The rod should be driven into the sand without rotating, and when the desired depth is reached it is rotated several times and the core withdrawn.

Another sampling rod is shaped similar to a screw auger. This is driven down into the sand with greater ease than the former, as it may be rotated in the direction of the screw, and the core is drawn up without rotating. Both rods bring up cores containing a larger proportion of the ore near the surface than near the bottom. This is a disadvantage which is more marked in the former than in the latter shape.

**Testing Laboratory and Fittings.**—The testing laboratory is preferably attached to the assay laboratory, and should consist of at least two separate but additional rooms, one for mechanical and the other for metallurgical work. The metallurgical room should contain apparatus for the leaching of sand, the settlement and agitation of slime pulp, and the precipitation of the gold

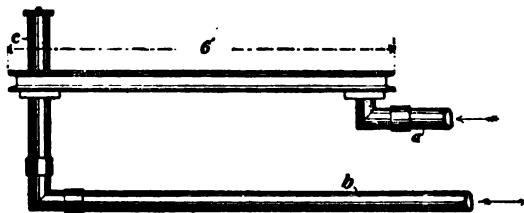


FIG. 8.—Hot Plate.

and silver from solutions. This office should also contain the usual supply of glassware, chemical reagents, and suitable balances.

The mechanical room should contain special apparatus for the comminution of the ore and the drying of samples. Ample space should be provided, and a concrete floor insisted upon.

It is without doubt false economy to under-man or under-equip the testing and assay laboratories, as even such a small increase in extraction as two or three grains per ton, which is likely to result from technical investigation, is generally more than sufficient to cover the laboratory expenses.

The sampling room should be well lighted and ventilated, and kept as free as possible from dust and draughts. Amongst the chief articles required are a sampling plate for moist ore made of 6 ft.  $\times$  3 ft.  $\times$   $\frac{1}{4}$ -in. steel, well planished and kept as free as possible from rust, and a hot plate of about the same size for semi-dry samples. This is made, as shown in elevation Fig. 8, of two plates  $\frac{1}{2}$  in. thick, riveted to a rim of 3-ft. channel iron and well caulked. The plate should be kept at a fairly uniform temperature, not exceeding 150° F. This may be conveniently done by heating the water in a boiler and circulating it by means of tubes—(a) the hot pipe and (b) the cold—in the usual way; (c) is the water feeder. The plates are placed at a convenient height to work at when standing, about 2 ft. 6 in. to 2 ft. 9 in. Several scoops, trowels, and large spatules should be available. A good supply of shelves and cupboards is essential.

**Method of Procedure.**—The selection of a scheme of treatment based on detailed experimental work and results is to be recommended in preference to a duplication of an existing plant arrangement, however successfully the latter may be operating. Each particular ore has its own peculiar characteristics, and a close study of its general properties and a knowledge of the distribution of the gold and silver are of the utmost importance. It is advisable that all experimental work should be done on the property so that local conditions may be duly taken into account. All such work should be carried out, using water, lime, etc., of a kind which would be subsequently available for actual treatment.

Should impurities be suspected, an analysis of the water supply should be made because free acid and organic matter in suspension cause much trouble in actual treatment, resulting in unexpected consumptions of cyanide and alkali, or interfering with the retention of the gold and silver in the pregnant solution before it reaches the precipitation plant.

Where it is possible to make use of ordinary milling plant in preliminary work, tests made under such conditions offer more reliable data. In the case of an ore where the isolation of a concentrated product seems advisable, working tests can be made. In this case samples should be taken for assay or other purposes. An analysis of the concentrate will probably be of value for future reference. A sizing test on the tailing, without regrinding, should be made with the idea of determining whether any portion of the ore may be classified and discharged as being too poor to pay for further treatment or regrinding.

In the absence of milling plant it will be necessary to carry out the preliminary grinding of the ore in laboratory apparatus. For this purpose a small breaker and also a small disc grinder are fitted up, and the ore is ground to a fineness which takes into consideration the improved extraction which will probably result with increased comminution, and the class of filter or press which will be available for the treatment of the ore pulp on a large scale.

By setting the grinder to crush at, say, 30 or 60 mesh, a sizing assay

test may be made for the purpose of determining the distribution of gold and silver in such ground product.

Where the presence of refractory minerals in injurious quantities has been indicated by preliminary examination, it will be advisable to have a sample of the ore sent away for concentration and analysis.

**Amalgamation Test.**—When an ore is to go through the amalgamation process it is usually crushed wet and amalgamated at the same time, leaving the tailing to be treated by cyanide. But there are cases where an ore may be advantageously amalgamated after cyaniding, as, for instance, when much of the coarse gold is coated with sulphide films, as is often the case when the gold is found associated with stibnite and other sulphides. Such gold has a deeper colour than usual and amalgamates only with great difficulty, but it often happens that these films are easily removed by cyanide, thus leaving the gold in a clean, amalgamable state. Amalgamation after cyaniding necessitates greater expense than when the operation is conducted after crushing, but it has the advantage that dry crushing can be adopted if found desirable.

It is usual, in testing the sample, to amalgamate before sizing, but the reverse order may sometimes be advantageously employed. The test in its simplest form consists in taking the weighed quantity of ore to be employed in the cyanide test and panning it with water in a prospecting pan, with an ounce or two of pure mercury. About 1 lb. of ore at a time is placed in the pan and panned in the usual way until the desired quantity is finished, and the whole of the tailing and water is washed into a vessel, preferably of enamelled iron or glazed earthenware. The mercury or amalgam is next carefully retorted, and this is conveniently done by putting it into a porcelain or smooth gold-annealing crucible, placed in a miniature retort over a regulated flame or in a muffle furnace.

The resulting bead is melted with or without fluxes, and if necessary parted, then weighed and the results tabulated. The whole of the tailing and water is saved for the cyanide test. When it is not required to know the quantity of free gold present, much time is saved by simply streaming the ore over a well-'set' amalgamated plate before proceeding with other tests. The result is a product similar to that of ordinary tailing from the amalgamation process.

**Sizing and Elutriation.**—Screen sizing is effected by passing a weighed portion of the material through a series of screens. The operation may be conducted with dry ore, or water may be used to assist separation. Wet screening does not always give the same result as dry screening, and there are cases where the method of arriving at result should be given with the analysis.

Most of the work done with laboratory screens is valueless and useless for comparative purposes. This is due to the fact that ordinary assayers' screens are made of wire of different diameters by different makers; and, although the number of openings per linear inch may be the same, the size of the ore particles which the screen may retain or pass may vary widely

with two different screens of the same nominal mesh. Apart from the question of strength, the size of the wire used in making a screen is of no significance whatever to the metallurgist. It is the size of the opening which is of importance.

A series of laboratory screens capable of universal acceptance and use would be of value. Various suggestions have been made to this end, which may well be considered in detail:

(1) In this case the number of holes per square inch is taken as the basis of the series, and the diameter of the wire is determined on the principle that the effective screening area per unit of surface is to be the same for all sizes. The standard series of the Institution of Mining and Metallurgy, of which details are given in Table II., is an example of this type. An irregular series of grains is produced, in which the sizes vary by no definite ratio. There is little to be said in favour of this system.

(2) In the second case the *area* of the opening is the basis of calculation, such areas decreasing in a fixed ratio. The linear dimensions of the openings decrease in a ratio which is equal to the square root of an arbitrary number which may be called  $N$ . The bulk of the grains decreases regularly in the ratio of  $2\sqrt{N^3}$ . The W. S. Tyler Co., of Cleveland, Ohio, U.S.A., have put on the market a special line of screen scale sieves of this type in which  $N$  equals 2. The openings in these decrease and increase in such a manner that the area of each sieve opening is twice that of the next finer and one-half that of the next coarser sieve; and 200-mesh screening, with an opening of 0.0029 in., is taken as the basis of calculation and manufacture. Table I. shows the class of screening made by this company and useful for metallurgical work.

An advantage in the use of a standard screen of this character lies in the fact that graphic illustrations of screen analyses can be made which are invaluable for comparative purposes.

TABLE I.—*The Tyler Standard Screen Sieves.*

| Opening in Inches<br>Ratio $\sqrt{2}$ or 1.414. | Opening in<br>Millimeters. | Mesh. |
|---|----------------------------|-------|
| .363  | 6.680                      | 3     |
| .185  | 4.699                      | 4     |
| .181  | 3.327                      | 6     |
| .093  | 2.362                      | 8     |
| .065  | 1.651                      | 10    |
| .046  | 1.168                      | 14    |
| .0328   | .833                       | 20    |
| .0232   | .589                       | 28    |
| .0164   | .417                       | 35    |
| .0116   | .295                       | 48    |
| .0082   | .208                       | 65    |
| .0058   | .147                       | 100   |
| .0041   | .104                       | 150   |
| .0029   | .074                       | 200   |

TABLE II.—*I.M.M. Standard Laboratory Screens.*

| Number of<br>Openings per<br>Linear Inch. | Diameter of Wire. |       | Aperture. |       | Screening<br>Area. |
|---|-------------------|-------|-----------|-------|--------------------|
|   | In.               | Mm.   | In.       | Mm.   | Per cent.          |
| 5   | .1                | 2.540 | .1        | 2.540 | 25.00              |
| 8   | .063              | 1.600 | .062      | 1.574 | 24.60              |
| 10  | .05               | 1.270 | .05       | 1.270 | 25.00              |
| 12  | .0417             | 1.059 | .0416     | 1.056 | 24.92              |
| 16  | .0313             | 0.795 | .0312     | 0.792 | 24.92              |
| 20  | .025              | 0.635 | .025      | 0.635 | 25.00              |
| 30  | .0167             | 0.424 | .0168     | 0.421 | 24.80              |
| 40  | .0125             | 0.317 | .0125     | 0.317 | 25.00              |
| 50  | .01               | 0.254 | .01       | 0.254 | 25.00              |
| 60  | .0083             | 0.211 | .0083     | 0.211 | 24.80              |
| 70  | .0071             | 0.180 | .0071     | 0.180 | 24.70              |
| 80  | .0063             | 0.160 | .0062     | 0.157 | 24.60              |
| 90  | .0055             | 0.139 | .0055     | 0.139 | 24.50              |
| 100                                       | .005              | 0.127 | .005      | 0.127 | 25.00              |
| 120                                       | .0041             | 0.104 | .0042     | 0.107 | 25.40              |
| 150                                       | .0033             | 0.084 | .0033     | 0.084 | 24.50              |
| 200                                       | .0025             | 0.063 | .0025     | 0.063 | 25.00              |

TABLE III.—*Screen Series with Linear Dimensions of Openings  
Decreasing in the Ratio  $\sqrt[3]{2} - 1.2599208635$ .*

| Linear Dimension of Opening. |         | Linear Dimension of Opening. |       |
|------------------------------|---------|------------------------------|-------|
| In.                          | Mm.     | In.                          | Mm.   |
| 1                            | 25.3995 | .03937                       | 1     |
| .7937                        | 20.1583 | .0313                        | .7937 |
| .6300                        | 15.9999 | .0248                        | .6300 |
| .5                           | 12.6998 | .0197                        | .5    |
| .3969                        | 10.0793 | .0156                        | .3969 |
| .3150                        | 7.9999  | .0124                        | .3150 |
| .25                          | 6.3499  | .0098                        | .25   |
| .1984                        | 5.0396  | .0078                        | .1984 |
| .1575                        | 4.0000  | .0062                        | .1575 |
| .125                         | 3.1749  | .0049                        | .125  |
| .0992                        | 2.5198  | .0039                        | .0992 |
| .0787                        | 2.0000  | .0031                        | .0787 |
| .0625                        | 1.5875  | .0025                        | .0625 |
| .0496                        | 1.2599  | .0020                        | .0496 |

(3) In the third case the *bulk* of the grain is taken as the basis of the series, decreasing in each grade by a constant ratio. In this case the linear dimension of the opening decreases in proportion to the cube root of an arbitrary number. A series of this kind has been proposed by J. T. Hoover<sup>1</sup> and H. Stadler<sup>2</sup> in which the chosen number is 2, i.e. each grain size has

<sup>1</sup> *Bull. Inst. Min. and Met.*, May 9, 1910.<sup>2</sup> *Trans. Inst. Min. and Met.*, 1908-10.

half the bulk of the preceding size. The linear dimensions of the openings decrease in the ratio  $\sqrt[3]{2}$ . An unpremeditated advantage in this series (see Table III.), mentioned by Hoover, is that by taking 1 in. as the largest size the 15th term of the descending scale is exactly 1 millimeter (0.03937 in.). This fact should facilitate the introduction of the series into countries using the metric system.

Satisfactory results in grading ores can be obtained by employing a small elutriation apparatus, as shown in Fig. 9, which consists of three cones, having an angle of about  $33^\circ$  at their apexes, and having diameters of bases in the proportion of 1 : 2 : 4. Other proportions are not of much importance. The cones are covered by hollow conical pieces, and fitted with wide mouths

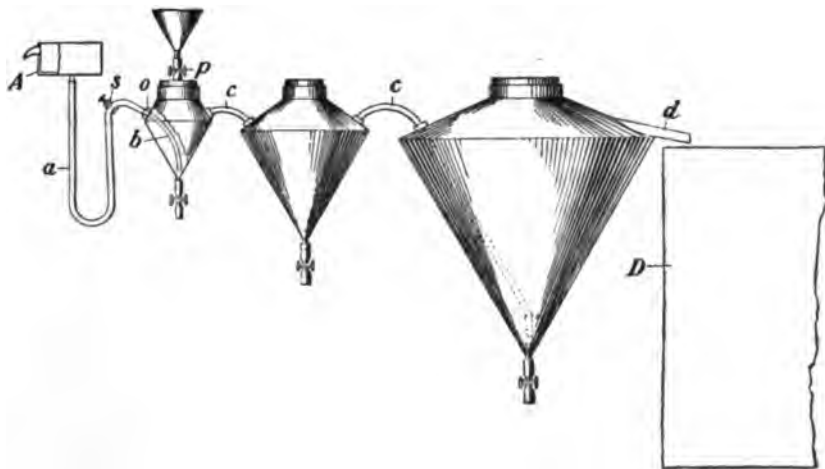


FIG. 9.—Elutriation Apparatus for Sizing and Classifying.

and rubber stoppers. In the stopper of the first cone is inserted a short glass tube, to which, by a rubber tube, is connected a funnel.

A is a clean water cistern, adjustable to different heights, and connected by means of a rubber tube *a* with a metal or glass tube *b*. This passes through a rubber stoppered tubulure *o* to the apex of the cone. A similar tube passes to the apex of each cone in the same manner, and a connecting tube *c* is inserted, as shown, to convey the water and particles of ore in suspension to the next cone.

The vessel *A* is adjusted to a suitable height and water is allowed to flow into it a little faster than it flows out through tube *a*, and is thus kept overflowing. When the apparatus is filled with water a clip *s* is closed and a small quantity of ore in the form of sludge is fed from the funnel into the first cone by loosening the spring clip *p*; *p* is then closed and *s* opened, and the water is allowed to flow from the cistern *A* until it is practically clear as it leaves the discharge pipe *d*. At the apex of each one is a short tube of rubber with a spring clip for the purpose of discharging the ore into separate

vessels at intervals. The operation is repeated until a sufficient quantity of each size is collected. The fine and light particles flow into cistern *D*, where they are allowed to settle. The clear water having been syphoned off, the residue is next dealt with by again repeating the operation, but with a much reduced pressure of water at *A*. The pressure is in fact reduced to  $\frac{1}{16}$  of the original.

In this way seven sizes of an ore may be obtained ; and if each size is submitted to a percolation test, the investigator can readily find out the proportion of non-leachable product present, and in what sizings the gold and silver chiefly exists. Other factors may also be rendered apparent, as, for instance, whether or not the ore has been crushed sufficiently fine to liberate the gold and silver ; and tests may be made on each size, by leaching with cyanide, to find out where the bulk of the metal remaining in residues is to be found.

**Mechanical Tests.**—It is not the intention to deal with the subject of crushing the ore, although much may be learned from the investigations as to which method of crushing will be the best to adopt ultimately.

The prepared sample may be delivered in the sampling room as rock, crushed ore, either wet or dry, or as tailing.

The first operation is to examine the ore under a low magnifying microscope, and determine as far as practicable the physical state of the gold and silver and the associated minerals. Much may be learned by an experienced man and time often saved from such an examination. If the gold appears to have a smooth homogeneous surface, or appears as well-defined crystals, it may be taken that the cyanide will, in all probability, dissolve only a small portion of it, and amalgamation will have to be resorted to if these gold particles are to be extracted. If, on the other hand, the gold is spongy or honeycombed in appearance, finely divided or invisible, it is possible it may be treated direct by cyanide. Then, again, the matrix may be of such a nature that crushing produces an excessive quantity of slime, as, for instance, when an ore is clayey or consists of certain schistose rocks ; in which case wet crushing might produce, for the most part, a product impervious to the solution, while with dry crushing, leaching may become practicable with a larger percentage of the ore ; or the ore may be quartz, and then either dry or wet crushing could be adopted. The decision would be influenced by the local conditions and the physical state in which the gold exists. When silver is present in a compound state, the microscope will often show which compound is acted on by the cyanide solution and which is not, and this information may lead to important results in treating silver ores.

The presence of metallic mineral matter, such as sulphides of iron, copper, antimony, zinc, etc., arsenides and tellurides, should always be determined by the microscope, their systems of crystallisation and the minerals identified, as far as practicable, in every unknown ore. This examination may lead to important results bearing on the method of procedure in treating an ore, and save much time in experimenting. Also in cases of failure it may indicate

the cause, and methods of overcoming difficulties may be thereby suggested. The microscope will also sometimes assist in determining whether or not the coarse particles of gold are amalgamable before or after cyanide treatment.

**Percolation Test.**—An important test is to determine what percentage of ore is capable of being treated in leaching vats and what quantity would have to be dealt with in other ways. The cost of leaching treatment is low as compared with other methods of handling, which invariably involve a finer comminution of the ore. If a satisfactory extraction can be obtained, it is highly advisable to treat the largest quantity practicable by this method. In the case of silver ores, leaching treatment has been uniformly unsatisfactory, and recourse is generally made to what is now known as all-sliming, or the reduction of the whole of the ore to a uniform fineness, followed by some method of agitation treatment. With such ores it is evident that the solution of a high percentage of the silver is impossible after only coarse crushing. This is doubtless due to the lack of exposure of much of the metal to the solvent action of the cyanide. To obtain a moderate extraction on ordinary silver ores it is necessary to allow for an abnormally extended leaching treatment, which often results in a premature precipitation in the lower parts of the vats. In view of the great saving of expense in the avoidance of excessive comminution in the case of gold ores, exhaustive tests should be made in an endeavour to find the economic grinding limit, the aim being to produce efficiency in extraction with economy in comminution and treatment. Gold and silver ores are either cyanided at rest or in suspension. The latter is by far the more expensive method.

The rate of percolation is determined sometimes by the time the solution above the ore sinks to a certain depth; thus, over 3 in. per hour is good,  $1\frac{1}{2}$  in. per hour is fair, and  $\frac{3}{4}$  in. per hour is bad. Less than  $\frac{3}{4}$  in. per hour is usually uneconomical. Louis Janin, jr., says<sup>1</sup> that tailings from the Bremen Mill, Silver City, New Mexico, ground fine in pan amalgamation, leached at the rate of  $\frac{1}{16}$  in. per hour, although aided by pump suction under the filter. He gives the rate for raw ores crushed dry by rolls through 20-mesh screen at 6 in. per hour as a minimum.

At Aspen, Colorado, in a Russell plant, the natural leaching rate was 13 in. per hour, reduced to 10 in., however, after the ore had been washed.<sup>2</sup> The rate of percolation varies widely with ores of the same fineness, being dependent largely on the method of charging the vats.

A better method than the above is to compare the porosity of the ore by comparing the quantity of moisture retained after draining. The greater the percentage of moisture retained, the smaller the porosity and the slower the percolation. In these tests the time allowed per foot of depth of vat should be that which could be economically given on a large scale.

In some classes of ore the amount of slime is so small and of such a character that, when mixed evenly with the leaching products, it does not

<sup>1</sup> *Mineral Industry*, vol. i. p. 253.

<sup>2</sup> Wm. S. Morse Prescott, *Trans. Am. Inst. Min. Eng.*, xxv. p. 143.



materially interfere with percolation or with the extraction of the metal. In such a case it might be most economical not to size at all.

To compare the porosity of the different sized products the apparatus shown in Fig. 10 may be employed. *A* is a tube of 1 to 2 in. diameter and 4 or 5 ft. long, connected at its lower end to a measuring glass *B* by means of rubber tubing *C*. The tube *A* is filled to a mark, say 4 ft. from the bottom, with the ore, previously dried, and the net weight noted. A rubber stopper *a* fitted with a short length of glass tube serves as a means of connecting *C*.

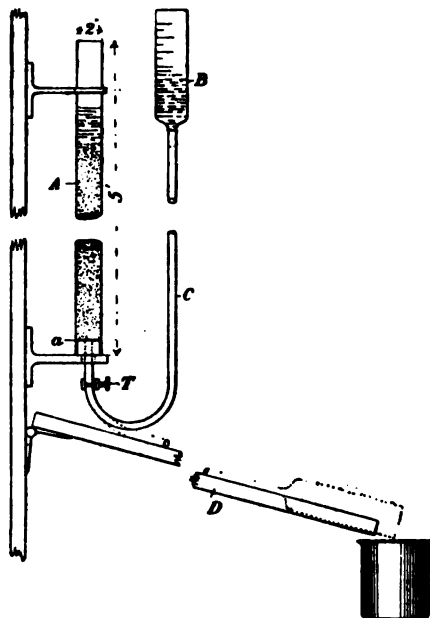


FIG. 10.—Apparatus for Testing the Porosity of Leaching Material.

The hole is covered over with a piece of muslin to support the sand. The glass *B* and tube *C* are next filled to zero with a clean cyanide solution of working strength, and the zero mark is brought on the level of *a*. The cock *T* is then opened and *B* is very gradually raised until the solution in *A* rises to a point about 3 in. above the ore surface and the solution in *B* is on the same level. *B* and *C* are next lowered and allowed to lie on a sloping board *D*, which may be adjusted to any angle between horizontal and vertical according to the suction found desirable, and then maintained in the same position for any set of experiments. Time observations may then be made of the fall of the solution in *A*, allowing the ore to drain for as many minutes per foot of depth as would be em-

ployed on a large scale. Finally, disconnect *C* from *a* and weigh *A* to find the weight of moisture retained in the ore. The less the moisture retained the more suitable is the ore for treatment by percolation in vats, leaving out, of course, the question of dissolution of the metal.

Table IV. shows the results on a sample of ore which had been sized so as to reduce the quantity of slime to a minimum.

This ore was crushed dry and consisted of schistose rock containing arsenical pyrite. It was found too impervious to treat direct in the ordinary leaching vats; but by close sizing 91 per cent. could be dealt with in this way.

"The great majority of estimations made to determine possible extraction by leaching are of little value on account of the absurdly small amounts of sand taken for treatment in the preliminary experimental work; and the production of data obtained from investigations on a depth of material too

shallow to demonstrate the chemical changes in the solution, and resultant effects on extraction occurring in an ordinary leaching vat. In many instances the residual content varies directly with the depth leached; and exceptional cases have been known where a residue sample taken from the bottom of a leaching vat, after prolonged treatment, has been found to con-

TABLE IV.

| No. of Size. | Per cent. of Total Weight of Ore Sample. | Weight of Ore Dry of Equal Bulks. | Weight of Ore after Draining. | Moisture retained per cent. of Ore. | Remarks.  |
|--------------|--|-----------------------------------|-------------------------------|-------------------------------------|---|
| 1            | 31.4                                     | 1800                              | 1881                          | 4.5                                 | } Percolated freely.  |
| 2            | 18.2                                     | 1455                              | 1556                          | 7                                   |   |
| 3            | 21.2                                     | 1605                              | 1755                          | 9.3                                 |   |
| 4            | 12                                       | 1226                              | 1367                          | 11.5                                | } Percolated 1 in. in 1½ hour.<br>Would not percolate in time allowed. Ore loss 0.5 per cent. |
| 5            | 8.2                                      | 1005                              | 1148                          | 14.2                                |   |
| 6            | 5  | 954                               | ...                           | ...                                 |   |
| 7            | 3.5                                      | 902                               | ...                           | ...                                 |   |

tain a higher metal content than did the original charge. Leaching tests are preferably made in lengths of ordinary piping, of a diameter consistent with the size of the sample available. The pipes are fitted with bored flanges, at one end for suspension and at the other for the filter-bottom. The latter is kept in place by a third flange with insertion joint, and fitted with a plug tapped for a leaching-control pipe, an arrangement incidentally providing for the upward charging of strong solution. The pipes should be cut or compounded of a length equal to the height of the vat it is proposed to use in practice: or, if the dimensions of the latter have not been decided upon, experiments should be made with various lengths. In all cases, and at the conclusion of treatment, bored samples should be taken at the top and also at the bottom of the pipe, and the assay results compared. Equality of residual content would indicate the possibility of treating larger bulks of ore by an increase in the height of percolation. A higher residue at the bottom than at the top would indicate the necessity for reducing the leaching depth, or resorting to double treatment. Head and tail samples should be taken after the charge has been thoroughly mixed, both before and after treatment."<sup>1</sup>

**Decantation Tests.**—These tests are usually made by thoroughly agitating known weights of slime and dilute cyanide solution with different percentages of some coagulating substance, allowing the mixtures to stand, and comparing the depths or volumes of the sediments. Ordinary cylindrical measuring glasses of 1000 c.c. may be used for these tests. Weigh off equal weights of slime, say five lots of 500 gm. each, and put each into a 1000-c.c. measuring glass together with different percentages of some coagulating substance, such as lime, say 0.1, 0.2, 0.4, 0.6, 0.8 per cent., add a dilute

<sup>1</sup> "Laboratory Cyanide Tests," A. W. Allen, *Min. and Sci. Press*, Feb. 17, 1912.

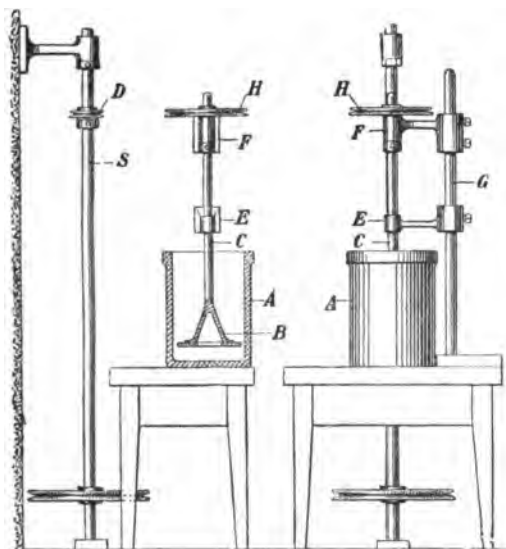
cyanide solution, say 0.05 per cent., a little at first, and thoroughly agitate; then fill up to the 1000-c.c. mark. Allow these to stand about three or four hours, taking care that the temperature remains constant, and note the depth of sediment in each case. The best percentage of lime is that which gives the greatest amount of clear solution for decantation. In some cases a maximum point is soon reached, whereas in others the most economical amount of lime has to be determined partly by cost. Beyond a certain amount of lime no advantage is derived, but often subsidence is retarded.

Caustic lime is sometimes employed in the cyanide process, as it protects the cyanide, but it is not so good a coagulating substance as other compounds of

calcium. Thus calcium carbonate (chalk) is often more rapid in its action and produces a more dense sediment, which is not so easily disturbed while decanting as when the hydrate is used.

**Agitation Tests.**—These are carried out on the slime which has been separated out from the crushed ore, or from the whole of the sample after it has been all-slimed.

The pulp may be agitated in a simple type of mechanical agitator such as that illustrated herewith (Fig. 11). *A* is an ordinary earthenware filter jar that contains an agitator *B*, made of a disc of iron or lead with a hole in



End Sectional Elevation.

Front Elevation.

FIG. 11.—Apparatus for Agitation Test.

its centre. It is secured to a spindle *C*, carried by two brackets *E* and *F*, and these brackets are secured by set screws to a stiff vertical pillar *G*, and may be raised or lowered. On the top of the spindle is keyed a pulley *H*, driven from a vertical shaft *S* by a pulley *D*. *D* is secured by a feather and collars, and is adjustable to any height of *H*.

For air-agitation experiments, a model Pachuca, or Brown, agitator is a necessary adjunct in any modern metallurgical laboratory. The vat can be conveniently and easily made of 23- or 24-gauge flat galvanised iron, folded at the joints, soldered, and afterwards given a coat of P. & B., or other paraffin paint.

Agitation tests should be made with pulp of varying thicknesses to determine the maximum specific gravity at which the pulp may be agitated in order to produce maximum solution of gold and silver. Care should be taken that the air used in such experimental apparatus is reduced in pressure

and volume to the lowest possible limit, otherwise excessive evaporation will occur.

The ratio of diameter of vat to diameter of air lift is generally about 10 or 12 to 1. The cone should be steep, and the dimensions of the vat should be consistent with the size of the samples available for the tests.

Head samples should be taken after the charge is in efficient agitation and after it has been ascertained that there is no settlement around the cone at the bottom of the vat. These samples should be carefully dried and assayed entire. The alkalinity of the solution should be brought normal before the addition of cyanide; and at the conclusion of the experiment the solution should be separated from the slime, and the gold and silver precipitated with zinc. Tail samples should be taken after 4, 8, 12, 16, or more hours' agitation, or until there is no further reduction in the metal content of the washed residue. Frequent tests should be made to determine the loss of cyanide and alkali.

As soon as results are available, showing the conditions under which maximum solution of the gold and silver takes place in the minimum of time, a further test should be made under these conditions of alkalinity, cyanide strength, thickness of pulp, and time of agitation, for the purpose of determining cyanide consumption. The solution used should consist of re-standardised barren solution, and in making such titrations it must be remembered that in solutions from silver ore treatment of high or average grade there is a considerable amount of cyanide locked up in the form of the double cyanide of silver and potassium (or sodium or calcium) which is not included in the result of an ordinary titration for total cyanide. Final titrations should always be made after the precipitation of the contained silver.

**Precipitating Apparatus.**—It is advisable to keep a small zinc box for use when required, and this should be made so that the compartments are capable of being increased or decreased in area. The velocity of the solution passing through the box may then be varied within wide limits, and be made independent of the rate of inflow.

An electrical precipitation box may be provided, arranged so that the velocity of the solution may be varied. For use with this apparatus a number of accumulators, ten or more, may be conveniently employed, and a voltmeter and ampere-meter are essential.

Zinc-precipitation tests on a small scale are usually not as satisfactory as on the working scale; but many important points present themselves in the laboratory which are not apparent on a large scale, and *vice versa*. On the other hand, precipitation by electrical methods gives, if anything, better results on a small scale than in a box of working size.

**Roasting Samples.**—This is best accomplished in a large-sized muffle on a shallow clay dish, preferably shaped to fit the bottom of the muffle, and having sides 1 in. deep. Roasting is commenced at a dull-red heat which is gradually increased to a cherry-red near the end of the operation. The ore requires constant but careful rabbling, especially at first. On no account

should it be heated sufficiently to melt the gold or to form particles of matte, for in the first-mentioned case the gold takes a spherical form and becomes extremely slow in dissolving in the cyanide solution, and in the second case the gold is hardly dissolved at all. A piece of asbestos packing or some clean sand placed under the roasting dish will be found of service in regulating the bottom heat.

The ore is weighed before roasting, and care must be taken that none is spilt during the operation. If weighed again after roasting, a loss occurs, due to the presence of volatile substances contained by sulphides, arsenides, or tellurides. This should be noted and tabulated in the results of an investigation, as the quantity of these substances present may have some bearing on the extraction.

The cost of roasting can only be estimated from operations on a large scale, and will naturally vary in different localities and with the style of furnace used.

## CHAPTER III.

### PRELIMINARY INVESTIGATIONS.

#### *Section II.*

**Oxidation of Ore Test.**—Much saving in alkali and cyanide may sometimes be ensured by oxidising the ferrous oxides to ferric oxides, by means of air or an oxidising agent, before applying the cyanide solution. An apparatus for testing samples by oxidising with air is shown in Fig. 12. *A* is an ordinary filter jar containing the sample of ore ( $\frac{1}{100}$  ton). This jar is connected by a glass tube with one mouth of a Wolff bottle *B*, and the other mouth is connected by an iron tube with a cask *C*, of 40 to 60 gal. The cask is placed at a level below the filter jar and is filled with water. The tap *D* is next opened and the water is allowed to escape at a known rate into a measuring vessel. A measurable volume of air is thus drawn through a known weight of ore in a known time. The ore is sprayed at intervals with a dilute alkaline solution of known quantity, which is caught in the Wolff bottle to be tested and the loss noted. A cyanide solution of 0.2 per cent. is then passed through the ore and the loss noted. This is compared with another sample of the same ore, treated direct with alkali and cyanide, and the costs in both cases estimated.

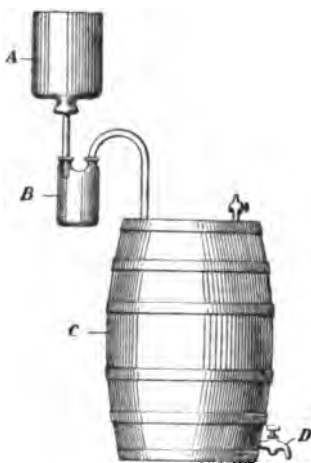


FIG. 12.—Apparatus for Testing the Oxidising Effect of Air on Samples.

Another method of reducing the consumption of cyanide is to run a dilute solution of an oxidising agent through the ore before adding the cyanide solution.

Sodium peroxide is a suitable oxidising agent for this purpose. If the ore is first treated with a dilute alkaline solution and afterwards with a dilute solution of an oxidising agent, as potassium permanganate or ferri-cyanide, a saving in cyanide is effected. The most economical strength of such solutions can only be found by experiment on ores with different strengths of solution, the results noted, and costs estimated.

When ore is treated by agitation the consumption of cyanide may be reduced by blowing air into the pulp at the centre of the tank near its

bottom, and adding at intervals small quantities of lime or alkali before adding the cyanide. Oxidising agents may also be used instead of air, small quantities being added at intervals before the cyanide.

**Comparative Values of Oxidising Agents.**—The following values for various oxidising agents are given by D. Mosher<sup>1</sup>:—

|  |       |                             |
|--|-------|-----------------------------|
| Sodium dioxide, $\text{Na}_2\text{O}_2$ . . . . .                    | 20.5  | per cent. available oxygen. |
| Barium dioxide, $\text{BaO}_2$ . . . . .                             | 8.0   | „ „ „                       |
| Hydrogen peroxide, $\text{H}_2\text{O}_2$ —8 per cent., . . . . .    | 1.5   | „ „ „                       |
| Potassium permanganate, $\text{KMnO}_4$ . . . . .                    | 12.7  | „ „ „                       |
| Sodium bichromate, $\text{Na}_2\text{Cr}_2\text{O}_7$ , . . . . .    | 17.9  | „ „ „                       |
| Calcium hypochlorite, $\text{CaOCl}_2$ , . . . . .                   | 7.4   | „ „ „                       |
| Ammon. persulphate $(\text{NH}_4)_2\text{S}_2\text{O}_8$ , . . . . . | 7.0   | „ „ „                       |
| 'Linde' oxygen, . . . . .  | 96–98 | „ „ „                       |
| Chlorine peroxide, $\text{Cl}_2\text{O}_2$ , . . . . .               | 81.0  | „ „ „                       |

**Velocity of Solution with Various Oxidisers.**—Ya. Mikhailenko and M. J. Mescheryakov<sup>2</sup> have made extensive tests by suspending gold in a 0.2 per cent. potassium cyanide solution saturated with air, and then determining the loss of weight under normal conditions and also in presence of other substances. These investigators found that the velocity of the solution is gradually diminished with the presence of H ions, an excess entirely arresting the reaction; OH ions do not accelerate the reaction and an excess of them has a retarding influence.

In neutral solutions the following substances in solution have no influence on the velocity: quinone;  $\text{Na}_2\text{SnO}_3$ ,  $\text{KBrO}_3$ ,  $\text{KIO}_3$ ,  $\text{KClO}_3$ ,  $\text{Hg}(\text{CN})_2$ ,  $\text{CuCl}_2$ . On the other hand the velocity of solution is accelerated by the presence of  $\text{KClO}_4$ ,  $\text{KMnO}_4$ ,  $\text{NH}_4\text{SO}_4$ ,  $\text{Na}_2\text{O}_2$ ,  $\text{KSO}_4$ ,  $\text{NaSO}_4$ ,  $\text{K}_3\text{Fe}(\text{CN})_6$ , and  $\text{KCO}_3$ .

If the accelerating influence of  $\text{KClO}_4$  is taken as unity, the value of the other substances is as follows:

|   |   |
|---|---|
| $\text{KClO}_4$ . . . . .   | 1 |
| $\text{KIO}_4$ or $\text{KCO}_3$ , . . . . .                            | 2 |
| $\text{NH}_4\text{SO}_4$ . . . . .                                      | 3 |
| $\text{KSO}_4$ , $\text{NaSO}_4$ , or $\text{Na}_2\text{O}_2$ . . . . . | 4 |
| $\text{K}_3\text{Fe}(\text{CN})_6$ , . . . . .                          | 5 |

The velocity of solution with these substances varies with concentration until a maximum is reached, after which a retarding influence on velocity is noticeable. By using a common factor, E, the maximum concentrations may be represented as follows:

|  |        |
|--|--------|
| $\text{NH}_4\text{SO}_4$ or $\text{Na}_2\text{O}_2$ , . . . . .                      | 0.02 E |
| $\text{K}_3\text{Fe}(\text{CN})_6$ , $\text{NaSO}_4$ , or $\text{KIO}_4$ , . . . . . | 0.04 „ |
| $\text{KSO}_4$ , . . . . .   | 0.10 „ |
| $\text{KCO}_3$ , . . . . .   | 0.20 „ |

The minimum amount capable of accelerating the reaction is, in most cases, 0.0004 E. The sum of the influences of several oxidising agents, when used together, is less than the maximum of the strongest of them.

<sup>1</sup> *Min. and Sci. Press*, May 15, 1909.

<sup>2</sup> *Jour. Russ. Phys. Chem. Soc.*, 44, 567–570; *Min. and Sci. Press*, Oct. 19, 1912.

**Effect of Bromo-Cyanide on Silver Ores.**—T. P. Holt<sup>1</sup> has shown that extraction is at a minimum with bromo-cyanide alone on silver ores; but that addition of bromo-cyanide in the ordinary method of cyanidation produces high extractions of the metal when the latter is in the form of argentite or native silver.

**The Cyaniding of Silver Ores after Roasting.**—The same investigator publishes results of experiments<sup>2</sup> and shows that high extractions by cyanide follow a chloridising roast of silver sulphide ore; whereas a low extraction follows an oxidising roast. The good results are, however, more than counterbalanced by the fact that a chloridising roast, in addition to the expense involved, results in a volatilisation loss which makes the method impracticable.

<sup>1</sup> *Min. and Sci. Press*, vol. 99, p. 160.

<sup>2</sup> *Ibid.*

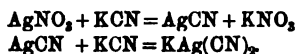


## CHAPTER IV.

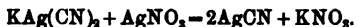
### THE TESTING OF CYANIDE SOLUTIONS AND ORES.

THE success of the cyanide process under wide variations of solution content has led, in many instances, to a disparagement of precise methods of testing. The chemistry of the process is, of course, a complicated study, but it is quite possible to draw sharp lines of demarcation between the chief substances found in ordinary working cyanide solutions; and it is imperative that the main reactions occurring during any test should be understood, otherwise an erroneous conclusion may result. If the reactions upon which the test is based are altogether ignored, it is evident that each operator may have his own opinion with regard to the carrying out of the test. Under such conditions similar investigations carried out by different operators may produce contradictory results; and any comparison, for economic or metallurgical purposes, is impracticable.

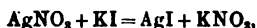
**Liebig's Test for Cyanide.**—In the early work in connection with the cyanide process this method was largely used to obtain accurate estimations of the percentage of free cyanide in the solution. In this method of testing, a standard solution of silver nitrate was slowly added to a measured quantity of the solution to be tested until a permanent turbidity ensued. The equations representing the reaction are as follows:



The white precipitate finally formed is due to the action of the first excess of silver nitrate on the potassium-silver-cyanide thus:



**Potassium Iodide as an Indicator.**—One of the earliest modifications introduced was found in the addition of potassium iodide to the solution to be tested, with the result that a more definite end-point was obtained. The silver iodide formed by the addition of  $\text{AgNO}_3$  dissolves in any free cyanide present; and the end-point, a yellow opalescence, is only observable after the whole of the cyanide present has been formed into potassium-silver-cyanide. The final reaction is, therefore, as follows:



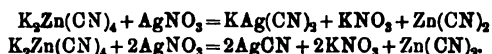
which occurs in precedence to a precipitation of silver cyanide.

The preferential action of silver nitrate for potassium iodide is of considerable value in the testing of cyanide solutions. In the case of a

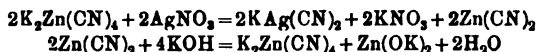
protectively neutral solution, containing zinc compounds, the yellow opalescent end-point is reached before any combination of silver nitrate and double cyanide takes place.

**Effect of Zinc on the Test.**—With the introduction of zinc as a precipitant and the addition of potassium-zinc-cyanide to the solution a number of complications arose, and Liebig's test gave erratic results. It was then found that the titration was affected by the quantity of free alkali present which proportionately increases the free cyanide content, as indicated by this method of testing. With an entire absence of evidence it was assumed that free cyanide was regenerated from the zinc-potassium-cyanide by the addition of alkali. This theory formed the easiest explanation of the phenomenon, and was generally adopted. Of later years, however, a number of cyanide chemists have found it increasingly difficult to accept a contention which, if carried to its logical conclusion, insists that, in an alkaline solution, gold and silver may be deposited in the zinc-box or press, and free cyanide entirely regenerated.

Both C. J. Ellis<sup>1</sup> and J. E. Clennell<sup>2</sup> have proved that the following reactions take place on adding silver nitrate to a solution of zinc-potassium-cyanide in absence of alkali :



In the presence of alkali, Ellis assumes that the following reactions occur simultaneously :



or expressed as one equation :



These equations offer evidence that a test of a working solution made by Liebig's method is liable to result in a faulty estimation of free cyanide, due to the combination of silver nitrate with zinc-potassium-cyanide.

In order to test the validity of Ellis's equation, a number of tests and experiments were made<sup>3</sup> on normal working cyanide solutions, the results of which go to prove that the addition of alkali under such conditions does not result in the regeneration of free cyanide ; and that the end-point in an ordinary titration test is due to the action of the silver nitrate on a combination of alkali and double cyanide.

E. M. Hamilton<sup>4</sup> has described a method of titration for free cyanide by direct addition of  $\text{AgNO}_3$ . The end-point, which the writer considers may be due to a precipitate of zinc cyanide or zinc ferrocyanide, is marked by the appearance of a bluish tinge dulling the original brilliancy of the solution.

<sup>1</sup> *Jour. Soc. Chem. Ind.*, Jan. 28, 1897.

<sup>2</sup> *Chemistry of Cyanide Solutions*, p. 37.

<sup>3</sup> "Solution Control in Cyanidation," by A. W. Allen, *Min. and Sci. Press*, Sept. 20, 1913.

<sup>4</sup> *Min. and Sci. Press*, July 25, 1914.

## SOLUTION CONTROL.

It is not the intention to go deeply into the question of the chemistry of cyanide solutions, or to deal with compounds of secondary importance or rare occurrence. The whole question has been ably dealt with by J. E. Clennell<sup>1</sup> in other publications to which the reader is referred for comprehensive information on the subject. It remains to describe, without unnecessary elaboration, the simplest and most accurate methods of testing ordinary cyanide solutions which contain only the usual impurities.

## PREPARATION OF TEST SOLUTIONS.

*Silver Nitrate.*—This solution is usually made by dissolving 6.523 gm. of pure recrystallised silver nitrate in one litre of water. The solution should be kept in a dark place, or the contents of the bottle completely covered from the light.

*Alkaline Iodide Indicator.*—This is best made by dissolving 10 gm. of potassium iodide and 40 gm. of pure caustic soda in a litre of water.

*Neutral Iodide Indicator.*—

|                  |   |   |   |           |
|------------------|---|---|---|-----------|
| Potassium iodide | . | . | . | 10 gm.    |
| Water to.        | . | . | . | 1000 c.c. |

*Phenolphthalein Indicator.*—One gm. of the salt is dissolved in 50 c.c. of absolute alcohol, and made up to 100 c.c. with water. Methylated spirit should not be used in place of the alcohol.

*Potassium Ferrocyanide.*—A 3 per cent. solution in water is generally used.

*N/10 Sulphuric Acid.*—Decinormal sulphuric acid contains 4.9 gm. of pure  $H_2SO_4$  per litre. The solution is made up of a slightly higher strength in the first instance and then standardised against a solution of pure sodium carbonate in the usual way. As in all other branches of analytical work, the necessity for absolute cleanliness of all glassware and water used is imperative.

**Total Cyanide.**—This estimation is used to determine the amount of cyanide present in the form of the simple cyanides of the alkalis and the alkaline earth metals, the double cyanides of zinc and alkali, and the hydrocyanic acid present. The test is made with silver nitrate solution. Potassium iodide is used as an indicator and the solution is made strongly alkaline with caustic soda.

The principal precaution to be observed in the titration for total cyanide refers to dilution. A certain amount of dilution is unavoidable, and it is sometimes found preferable to add a definite amount of distilled water for each test. Pipettes used for measurement should be washed out, and, by using the same amount of water each time, uniformity of condition may be ensured together with comparative accuracy of result.

It is usual to take 10 c.c., or any multiple thereof, of the solution to be tested, to which is added a measured quantity of alkaline iodide indicator.

<sup>1</sup> *The Chemistry of Cyanide Solutions. The Cyanide Handbook.*

The silver nitrate is then slowly added from a burette until a distinct yellow coloration forms and persists.

Each c.c. of the silver nitrate solution added to 10 c.c. of the original cyanide solution represents 0.05 per cent. total cyanide in terms of potassium cyanide. This is generally recorded thus: 0.05 per cent. (KCN).

The total cyanide estimation, as outlined, forms the simplest and most reliable method of estimating the available and partly available cyanide content of an ordinary solution. The end-point is definite and unmistakable, and the estimation is quickly and easily made. Its value as an indication of the 'strength' of the solution is seen in all cases where zinc content remains constant; and, under these circumstances and for comparative purposes, the result is preferable to a free cyanide estimation made by methods of dubious validity.

**Protective Alkalinity.**—The protective alkalinity of a cyanide solution is the equivalent of the amount of dilute acid required to effect neutralisation of the solution without decomposing the cyanide present.

In the absence of zinc this may be estimated by Clennell's<sup>1</sup> method; and 10 c.c. of the solution, or any multiple thereof, may be taken, as in the previous test. Silver nitrate solution is then added until a precipitate forms. Phenolphthalein is used as an indicator and the mixture is titrated with N/10 acid. Each c.c. of acid solution required to neutralise 10 c.c. of the original solution represents 0.04 per cent. protective alkali, in terms of caustic soda. This is generally recorded as 0.04 per cent. (NaOH).

In the presence of zinc compounds in the solution, Green's modification should be introduced into the test. This consists of the addition of excess potassium ferrocyanide solution before silver nitrate is added. The test is then carried out in a similar fashion to the one previously described. In both cases care should be taken that the acid is added drop by drop.

**Free Cyanide.**—The free cyanide estimation is used to determine the amount of cyanide present in the form of cyanides of the alkalis and alkaline earth metals. In the absence of zinc it may be made by Liebig's method already described. It is usual to use potassium iodide as an indicator, and the absence or presence of free alkali does not interfere with the result.

It has been shown that, in the presence of zinc, any free alkali present will react with the double cyanide on the addition of silver nitrate. This postponement of the end-point will result in a misleading conclusion as regards free cyanide content. In order to obtain a correct estimate it is preferable to reduce the solution to neutrality as far as protective alkalinity is concerned. This is best done by taking a fresh portion of the solution and by slowly adding the exact amount of acid shown to have been required in the protective alkalinity test. A few c.c. of neutral potassium iodide solution are then added and the mixture titrated with standard silver nitrate solution.

Each c.c. of the silver nitrate solution added to 10 c.c. of original solution in order to produce a yellow opalescent end-point represents 0.05 per cent.

<sup>1</sup> *Chem. News*, vol. lxxi, p. 98.

free cyanide, in terms of potassium cyanide. This is generally recorded as 0.05 per cent. (KCN).

The above estimations indicate methods by which routine testing can be carried out in the great majority of instances in connection with the cyanidation of gold and silver ores. In exceptional cases, and where impurities in quantity other than zinc are present, special methods of testing have to be evolved and additional determinations carried out.

The custom of reporting the cyanide strength of a working solution in terms of potassium cyanide dates from the early days of the industry. Later, varying quantities of sodium cyanide were incorporated with the potassium salt, often for the purpose of increasing the cyanogen content, and in order to maintain an average solvent effectiveness. Whatever the actual composition of the material, the estimation 'in terms of potassium cyanide' aided in the standardisation of data. It was assumed that all reports of cyanide content were estimated on this basis, although the fact was not always stated. Metallurgists the world over welcomed a system of recording strengths of solution which facilitated prompt and accurate comparison with the work in other plants and other countries.

The potassium cyanide standard was an arbitrary one, but it was entirely acceptable and logical. It might, of course, be argued that the best method would be to estimate 'in terms of cyanogen,' but nothing of advantage would have been gained. The value of all such figures lies in their availability for comparative purposes, and so long as only one system is used it is immaterial as to what compound is taken as the unit.

Early in 1916 an important American chemical company changed the designation of their cyanide products, and commenced to report commercial sodium cyanide in terms of the pure sodium salt. This was a logical procedure from the commercial point of view, but far from unobjectionable from the technical point of view; for the extension of the practice has led to general confusion, without any material gain in convenience and none in accuracy. A new reform will doubtless be instituted as soon as it is realised that the cyanogen in a working solution is not entirely in the form of the sodium salt; and it is probable that the estimation will then be made 'in terms of CN.' In the meantime a number of cyanide metallurgists are disinclined to discard the practice of reporting 'in terms of potassium cyanide,' realising that it has the advantage of uniformity, accuracy, and world-wide acceptance.

In advocating the adoption of the sodium cyanide standard W. J. Sharwood<sup>1</sup> has pointed out that it is possible, as the result of a convenient coincidence, to use N/10 or N/20 silver nitrate solution without necessitating any calculation. One cubic centimetre of N/10 silver solution is equivalent (by Liebig's titration, or using the preferable modification with potassium iodide indicator) to 5.202 mgm. of CN, or to exactly 9.802 mgm. of NaCN. Now 98 is almost the exact percentage of actual NaCN in the high-grade

<sup>1</sup> *Jour. Indus. and Chem. Eng.*, April 1, 1918.

commercial material now in use. Therefore we can titrate solutions with  $N/10$   $AgNO_3$  and call 1 c.c. equivalent to 10 mgm. of the actual 98 per cent. salt, which has to be added in standardising the solutions. For technical purposes it is perhaps preferable to use  $N/20$  solution (1 c.c. = 5 mgm. commercial  $NaCN$ ), as the end-point with iodide indicator is very delicate, and the burette readings then also indicate pounds per ton of solution directly. For instance, taking a 10-c.c. sample: suppose 2 c.c. of  $N/20$  silver nitrate is consumed; this indicates 10 mgm. or 0.10 per cent. of commercial sodium cyanide in solution, or 2 lb. per ton of solution.

**Estimation of Acidity in Ore.**—The amount of alkali which will be needed in practice may be estimated by preliminary tests. A quantity of the ground ore, the dry weight of which is known or can be subsequently estimated, is agitated with an excess of caustic soda solution. The latter is made up to carry a definite amount of alkali, and a portion is tested beforehand with decinormal sulphuric acid. After agitation and settlement for a time approximating actual conditions of practice, a portion of the solution which has been in contact with the ore is filtered off and again tested with the decinormal acid. The total acidity in the ore may then be calculated from the difference in the alkalinity estimations, and reported in terms of pure caustic soda or lime.

**Preliminary Estimation of Cyanide Consumption.**—The following precautions should be observed in any preliminary estimation of cyanide consumption, and regard should be paid to the question as to whether milling in water or milling in cyanide solution is to be adopted.

*Test to forecast Cyanide Consumption after Milling in Water.*—A weighed quantity of the ground ore is first treated with a sufficiency of lime water until no further consumption of alkali takes place. The moist ore, which should have a distinct alkaline reaction, is then agitated with a definite amount of dilute solution of known cyanide content. The strength of the solution used should be approximately the same as would be suitable in practice. The alkalinity of the solution should be maintained, if necessary, during the test by the addition of free alkali. The actual weight of cyanide brought into contact with the ore in the first instance is calculated from the weight of solution added and a cyanide titration percentage result. The actual weight of cyanide remaining after the test is calculated by means of an additional test to determine final cyanide content percentage and the actual weight of solution associated with the ore. From these figures the total cyanide consumption per ton may be calculated as a fraction or multiple of pounds of potassium cyanide per ton of ore.

*Test to forecast Cyanide Consumption after Milling in Cyanide.*—This test is carried out by agitating a known weight of the pulverised ore with a sufficiency of alkaline cyanide solution of known cyanide content. From the difference in the titration result before and after the test the probable consumption of cyanide per ton treated may be forecasted.

The above methods for testing for cyanide consumption are only available

in the case where the ore is a simple one and of low metal content. In high-grade ores, especially those met with in silver cyanidation, regard must be paid to the amount of cyanide which is associated with the metal dissolved, and which does not appear in the result of an ordinary titration test. In the cyanidation of silver ores or concentrates this may amount to from 1 to 30 lb. of cyanide per ton of material being treated. It is, therefore, advisable to precipitate the silver before the final titration is made. If aluminium is used as the precipitant then the simple cyanide test will give a reliable estimate of the untouched cyanide as well as the cyanide which has been regenerated during precipitation. In this case the actual consumption may be forecasted with tolerable accuracy.

If, on the other hand, the gold and silver is precipitated with zinc, the final total cyanide titration will include the amount of cyanide associated with zinc in the form of the double cyanide. It is highly probable that cyanide in such form is at least partly available for subsequent treatment, but it is certain that it is not available for the purpose of providing the excess cyanide needed for subsequent precipitation. It is, therefore, advisable to repeat the test more than once, using the precipitated solution over again on a fresh portion of ore and restandardised with the addition of cyanide. The consumption should be based on this latter amount and should be checked by the difference in titrations before and after the test. The major consumption of cyanide, when zinc accumulates in the solution, often takes place in the extractor plant, and this fact should not be overlooked in the preliminary investigations.

## CHAPTER V.

## PRINCIPLES INVOLVED IN THE DISSOLUTION AND PRECIPITATION OF METALS.

**Diffusion and Osmotic Pressure.**—If a vessel is taken which is divided by a porous membrane into two compartments, both containing water at different levels, the two liquids will in time adjust themselves until they are both at the same level. If next a substance is dissolved in one compartment, such as cannot easily pass through the pores of the membrane, it will be found that part of the water in the other compartment forces its way into the solution of the dissolved substance, until both liquids remain at rest at different levels. This phenomenon is termed *osmosis*; and the pressure to which this change is due, *osmotic pressure*. Osmotic pressure plays an important part in the cyanide process, both in the dissolving and precipitating of the gold and silver.

Pfeffer investigated this subject, and succeeded in preparing suitable membranes by which he made it possible to determine quantitatively the osmotic pressure of a number of chemical substances at different concentrations and temperatures. It has been proved in some cases that at the limiting surfaces between the water and the dissolved substance this pressure amounts to over a thousand atmospheres.

Van't Hoff, who had interested himself in Pfeffer's work, published a volume in 1887, showing that there existed a relation between the gas pressures of gases and the osmotic pressure of solutions. That is to say, that the well-established gas laws of Boyle, Gay-Lussac, Avogadro, and others also apply to the osmotic pressure of solutions. These laws are usually expressed by the equation

$$pv = RT \quad (1)$$

where  $p$  is the pressure and  $v$  the volume of the gas, while  $T$  denotes the absolute temperature and  $R$  is a constant for a perfect gas. There are, however, a number of exceptions in which the osmotic pressure does not obey the gas laws, and these are of most interest in the cyanide process. The exceptions include salts, acids, and bases; and in order to apply the above equation Van't Hoff introduced a coefficient  $i$  which modified the expression to

$$pv = iRT \quad (2)$$

If a salt such as potassium chloride be dissolved in alcohol, equation (1) holds good, and the coefficient  $i$  does not come into account; but if the salt is dissolved in water, the solution behaves as if a greater number of molecules were present than the concentration warrants. Thus, by halving the con-



centration, the osmotic pressure was always greater than half. It is concluded that this is due to a dissociation or breaking asunder of some of the molecules which occurs in aqueous solutions, but not in alcoholic solutions.

**Electrolytic Dissociation.**—Clausius<sup>1</sup> put forward a theory that when a salt such as NaCl was dissolved in water, some of the molecules broke down into ions; that is to say, while some of the molecules remained NaCl, a certain portion, as it were, broke asunder and became Na and Cl ions, and these ions were free to move in all directions as if they were independent molecules. Now, if the osmotic pressure was greater than it should be in accordance with the gas laws, it would appear to point to a greater number of particles present than assumed, such as the theory of Clausius suggests.

The conception of the independence of certain parts of the dissolved molecules was, up to this point, only of a qualitative nature. In 1887 Arrhenius put forward methods in two distinct lines of research, by which he was able to measure what percentage of the molecules became dissociated, and offered an explanation which accounted for the exception in Van't Hoff's generalisation.

The one method was based on the electrolytic conductivity of the solution, and the other on the thermodynamic theory of osmotic pressure, both methods giving results that agreed within limits of experimental error. Also it was soon found that only those substances that show abnormally high osmotic pressures conduct a current. From this it may be inferred that only those solutions are conductors that are, at least in part, dissociated into ions.

Arrhenius called undissociated molecules inactive, and those which had broken down into ions, active; and concluded that only active molecules or ions take part in the conductivity of the current; and further, that when a solution was highly diluted, it became completely dissociated into active molecules or ions. These ions differ from atoms and molecules inasmuch as they carry enormous electric charges, the one, *e.g.* Na, being positive, and the other, Cl, being negative, whereas atoms and molecules are electrically neutral. NaCl dissolves in water as molecules; but if this compound be decomposed in the presence of water into Na and Cl molecules or atoms, they instantly decompose the water and form new compounds. On the other hand, if NaCl is dissociated into ions, no such action takes place.

**Electrolytic Solution Pressure.**—Nernst conceived the idea that there was an analogy between the dissolving of a metal in a solution and the evaporation of a liquid into space. At a given temperature, if a solution is evaporated, it continues to pass into space until its vapour exerts a certain definite pressure called the vapour pressure, when equilibrium is established.

It may be assumed that, according to Van't Hoff's theory, the molecules of a dissolved substance exist under a definite pressure, which may be ascribed to the dissolving substance in contact with the solvent; further, that they have a power of expansion, being driven into space, where they exist under a certain pressure. Then, just as a liquid passes into space until the vapour in

<sup>1</sup> *Pogg. Ann.*, ci. p. 333, 1857.

the space exerts a pressure equal to the vapour pressure, so a substance dissolves in a liquid until the osmotic partial pressures of the molecules equal the solution pressure of the substance.

Nernst put forward an explanation, based on his theory, to account for the action of primary cells, from which it may be concluded that the seat of electromotive force is chiefly at the surface of contact between the electrodes and the electrolytes. When a bar of zinc is immersed in an acid, some ions pass into solution, and in order to do so they must take positive charges with them. The metal then becomes negative, and the solution being positive from the ions it has received, the so-called electrical double layer of Helmholtz is formed at the plane of contact, which resists any more ions coming into the solution. That is, the metal stops dissolving, although the amount dissolved is so small that it cannot be detected analytically. If a negative metal such as lead be immersed in the same solution, it immediately becomes covered with the positive zinc ions, which charge it with positive electricity. This proceeds until a certain potential is attained, when equilibrium is established.

Let the zinc and lead be touched momentarily. The positive electricity on the lead flows by the metallic conductor to the zinc, which had been negatively charged. The zinc being now positively charged, a fresh layer of zinc ions are again sent into the solution, which again leaves the zinc negative and charges the lead positively. If this operation is repeated a number of times it will be found that the zinc bar has lost weight, and that zinc has accumulated in the solution. In this way the metal dissolves, but if the two metals are kept in contact the process is continuous.

It is evident from Nernst's theory that the metal cannot send as many ions into a salt of zinc as into the pure acid, because the osmotic pressure of the zinc ions already in solution opposes any further ions being sent in, and therefore the solution pressure is lowered. Hence a metal becomes less soluble as metallic ions accumulate.

When the osmotic pressure of metallic ions in a solution is greater than the solution pressure of the metal, the metallic ions give up their positive charges and separate as atoms, *i.e.* precipitate. The solution now becomes negative, due to loss of some of its positive charges, and the double layer again occurs at the surface of contact between the metal and the solution, but this time in the reverse order; that is, the solution is negative and the metal positive. The processes of dissolution and precipitation are therefore closely allied and act in opposite directions.

**Anions and Cations.**—When a current is passed through a salt solution, that part where the current enters the solution is called the *anode* or *positive electrode*, and the part where it leaves the solution is called the *cathode* or *negative electrode*. The ions of the salt are directed by the current and are caused to move, the negative ions or anions going to the anode, and the positive ions or cations to the cathode. This action of the current is used to determine what part of a salt is the anion and what part is the cation. For simple

metallic salts the metals are the cations and the remainder, the anions. In bases, the hydroxyl is the anion and the remainder, the cation ; and in acids, hydrogen is the cation and the remainder, the anion. With complex compounds, however, these simple rules do not apply. Thus, for instance,  $K_4Fe(CN)_6$  dissociates into  $K_4$  as the cation and  $Fe(CN)_6$  as the anion ; and  $KAu(CN)_2$  into  $K$  and  $Au(CN)_2$ . Iron and gold in simple salts are always cations, whereas in complex cases they form part of the anions. These examples illustrate that what may be cations under one condition may be anions under another.

Ions (anions and cations) may carry one, two, three or more charges of electricity, and are usually, from this property, spoken of as univalent, divalent, trivalent, and so on. Ions of the same chemical composition do not necessarily carry the same number of charges, but may under one condition be univalent, and under another, divalent, and so on. Thus, in  $KMnO_4$ ,  $MnO_4$  is the anion, which is univalent, but in  $K_2MnO_4$ ,  $MnO_4$  the anion, in this case, is divalent. Also in  $K_4Fe(CN)_6$  the anion  $Fe(CN)_6$  carries four charges, but in  $K_3Fe(CN)_6$  the anion, although still  $Fe(CN)_6$ , carries only three charges. Consequently the properties of the two compounds are different.

## CHAPTER VI.

### DISSOLUTION OF THE GOLD AND SILVER.

#### *Section I.*

It is well known that there are great differences in the solubility of different metals in the same solvent, and of the same metal in different solvents, but no satisfactory explanation has been put forward why these differences exist, and until this is done the process of dissolving metals must be regarded as imperfectly understood. Thus it is known that platinum and iron are but slowly dissolved in potassium cyanide, whereas gold and copper are rapidly attacked. Also, gold dissolves in chlorine water, whereas it does not dissolve in nitric acid; and silver dissolves in nitric acid, whereas it does not dissolve in chlorine water.

What really causes a solution to dissolve one metal in preference to another is difficult to explain; but, to answer the purpose in discussing the solvent action of a cyanide solution on gold and silver, the hypothesis may be assumed to be correct that to dissolve a metal a difference of potential must exist between two parts in electrical contact with its surface and in contact with the solvent. It is well known that a pure metal is less soluble in a solution than similar metal of commercial quality; and it is assumed that if the metal is perfectly pure and has a surface of uniform density it would not dissolve in a solution of uniform concentration, because it would then have no difference of potential at any two parts. It would then be said that the solution pressure of the metal was the same at all parts of its surface; and in order that the metal may dissolve some alteration must take place, so that the potential at two parts of its surface may differ.

Two conditions are possible to produce a difference of potential between metals or metallic substances: (1) different substances, or the same substance of different density in electrical contact in the same solution; (2) the same substance of uniform density in different solutions, or the same solution in different states of concentration (strength).

The first case is the main source from which a difference of potential is derived in cyaniding the ore, by which gold is caused to dissolve and electric currents are produced.

Gold is never found in nature in a pure state, but is always alloyed or in intimate contact with other metals, and is generally in contact with some substance electro-negative in cyanide solutions to gold and silver, such as iron or arsenical pyrites. In the case of a crystal of pyrite, *A*, Fig. 13,

to which is attached a piece of gold *B* immersed in a solution of potassium cyanide, the gold, being electro-positive to the pyrite, dissolves. The dissolving action produces electric currents, which are made possible only by the presence of a negative electrode, *e.g.* the pyrite surface. The gold is therefore the anode of a simple cell, and the pyrite is the cathode. The currents take an innumerable number of paths, as *a, c, b* through the solid electrode, and *b, d, a* through the solution, as shown by the arrows.

In the second case, with a piece of gold not in contact with pyrite or other negative solid, but in contact with different solutions, or in solutions of different strengths: If *A, B*, Fig. 14, represents the piece of gold, of which the end *A* is in a strong solution of cyanide and *B* in a weak solution, the part in the strong solution is of a different potential from that in the weak solution. It will be found that the end *A* dissolves, and is therefore the anode, and the currents formed are led out of the solution at the end *B*, or cathode.

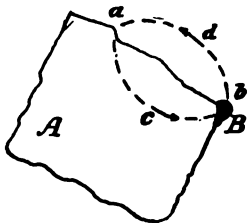


FIG. 13.—Auriferous Pyrite in a Cyanide Solution.

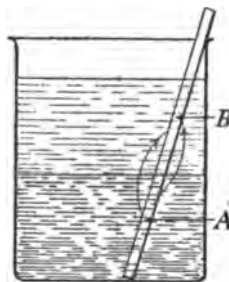


FIG. 14.—Gold in Cyanide Solutions of different Strengths.

The conditions in this case do not obtain, in treating ores by the cyanide process, to anything like the same extent as in the former case.

In both instances it is a question of short-circuited cells, identical with those formed in the process of dissolving gold and silver as contained in ores; but it would be impossible to make satisfactory investigations of the changes that occur while the two electrodes are in contact in this way. If, however, the anode and cathode are separated and short-circuited, by means of wires brought to the outside of the solution, with a suitable galvanometer intercalated, electrical and chemical changes can be readily and accurately noted.

**Dissolving Investigations.**—Fig. 15 shows similar conditions to those in the first case cited above, except that the pyrite *A* and the gold *B* are separated and connected through a galvanometer *G* by means of wires conducted outside the cyanide solution. With electrodes of the same size, the chemical and electrical changes are obviously identical, except perhaps when an alteration occurs in the resistance of the circuit, and the rate of chemical action is varied proportionately. When contact is made between the electrodes and the galvanometer, the deflection indicates that a current flows from *A*

to *B* through the wires, and thence through the solution from *B* to *A*. The measurement of the current strength provides data from which the quantity of metal dissolved can be determined, and results compared with any change in the conditions.

**Electro-chemical Changes.**—It is assumed that the KCN in solution is ionised, either wholly or partly, into K cations and CN anions, and that only ions take part in the dissolving action of the solution on the metal. When the current flows from the gold into the solution it takes positive charges, under the influence of the CN anion that conducts the current into the solution, and negative electricity flows away through the connecting wires. At the same moment, K cations reach the pyrite, giving up their positive charges. The gold, as attacked, apparently forms AuCN, and this is dissolved by the diffusion of an excess of KCN present to form  $\text{KAu}(\text{CN})_2$ . The K atoms separated at the pyrite cannot remain in a free state in the solution, but instantly attack the water, liberating H and forming KOH. The hydrogen becomes occluded either wholly or partly by the pyrite electrode, and to some extent interferes with the further dissolution of the gold.

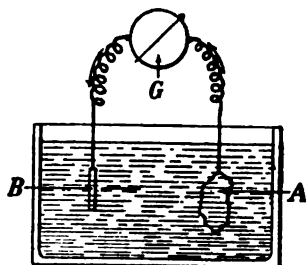
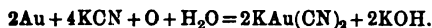


FIG. 15.—Gold in contact with Pyrite in Cyanide Solution.

**The Function of Oxidisers.**—The value of oxygen or oxidising agents has been much discussed, and although a few have disputed the necessity of oxygen for dissolving gold in cyanide solutions, the consensus of opinion is that some form of an oxidiser is essential for the successful carrying out of the cyanide process.

Elsner<sup>1</sup> stated in 1846 that gold is only dissolved by a cyanide in presence of oxygen, and at a later date an equation was given in Watt's *Dictionary of Chemistry* to meet Elsner's views as follows:—



J. S. MacLaurin<sup>2</sup> has made some interesting investigations on the effect of oxygen in dissolving pure gold plates in a solution of potassium cyanide, and he has come to the conclusion that—(1) "Oxygen is necessary for the dissolution of gold in potassium cyanide, and no gold is dissolved in its absence"; (2) "The ratio of the gold dissolved to the oxygen required for its dissolution is 196 : 8, as demanded by the equation" cited above.

W. Bettel<sup>3</sup> floated gold leaf on a cyanide solution and excluded oxygen by a continual passage of a stream of hydrogen, when no gold dissolved. He then tried the effect of the addition of different oxidising agents, as  $\text{K}_3\text{Fe}(\text{CN})_6$ ,  $\text{KMnO}_4$ ,  $\text{N}_2\text{O}_2$ ,  $\text{H}_2\text{O}_2$ , Cl, and Br, when the gold dissolved, more or less, rapidly.

<sup>1</sup> *Erdm. Jour. Prak. Chem.*, vol. xxxvii. pp. 441–446, 1846.

<sup>2</sup> *Jour. Chem. Soc.*, vol. lxviii. pp. 199–212, 1895.

<sup>3</sup> *S. African Min. Jour.*, May 8, p. 666, 1897.

J. S. MacArthur<sup>1</sup> says, "I never could find that the presence of oxygen was necessary, either to dissolve gold by itself or from ores by cyanide." He says he has seen no evidence that oxygen is absorbed in the dissolving of the gold, and has no proof that hydrogen is evolved, but thinks the latter the more probable.

S. B. Christy<sup>2</sup> says that his investigations confirm the accuracy of the reaction attributed to Elsner, and asserts that "pure cyanide of potassium in pure water from which all other substances are excluded is entirely without action on metallic gold."

Julian and Smart made some investigations on this subject, dealing with working cyanide solutions from which oxygen has been extracted, and gold as it exists in ores in contact with pyrites, and found that—

1. Gold and silver are capable of dissolving at ordinary atmospheric temperatures and pressures to a limited extent, without the addition of oxygen or oxidising agents.
2. For the continued dissolution of gold and silver at ordinary atmospheric temperatures and pressures, oxygen, or an oxidising agent, is desirable if not absolutely essential.
3. At ordinary temperatures the atmospheric pressure has an indirect influence on the constancy of the dissolving action.
4. Temperature has a direct influence on the dissolution of the metal, independent of the oxygen absorbed by the solution.
5. The value of oxygen absorbed by the solution is dependent on atmospheric pressure, and to some extent on temperature.

The following experiments illustrate the action. In Fig. 16 is shown a similar apparatus to that already described. *A* is the pyrite and *B* the gold

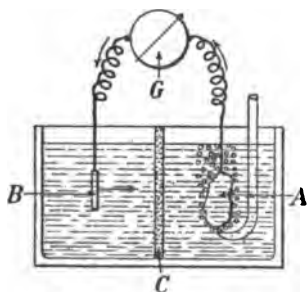


FIG. 16.—Gold and Pyrite in Cyanide Solution and the Effect of Oxygen.

in a working cyanide solution, and these are connected by insulated wires to a galvanometer *G*. As has already been shown, a current is generated by this combination, flowing from the pyrite through the galvanometer to the gold, and thence through the solution to the pyrite again, and so on. Before making contact with *A* and *B* the galvanometer needle is brought to zero, and immediately after contact the deflection is noted. It will be observed that the needle returns again towards zero, at first rapidly and afterwards more slowly, until a point is reached when the needle remains nearly

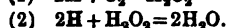
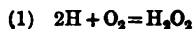
constant. Thus, in a case where the needle deflected to 20 divisions on the scale at the moment of contact, it returned in a few minutes to 10, and in half an hour to 4 divisions, showing that the current was reduced to one-fifth, and showing also that the rate of dissolution of the gold was reduced in like

<sup>1</sup> *Jour. Soc. Chem. Ind.*, vol. ix. pp. 267-270, 1890.

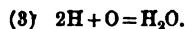
<sup>2</sup> *Trans. Am. Inst. Min. Eng.*, Sept. 1896.

proportion. If, now, a porous partition *C* be placed in the vessel in order to prevent the solutions surrounding the different electrodes from mixing, and if air or oxygen be blown into the solution at the gold surface *B*, little or no change takes place in the current; but if the air or oxygen be applied at the pyrite surface *A*, a sudden increase in the current takes place, and the needle returns to its original position of 20 divisions or thereabouts, showing that the gold is again dissolving at the same rate as when contact was first made. From this simple experiment it may be surmised that oxygen has no direct action in the dissolution of the gold. From the results of a number of experiments it may be concluded that, at ordinary temperatures and pressures, oxygen plays a secondary part only, in the dissolving of the metals.

What really happens is, that through the dissolving of the gold the current set up deposits hydrogen on the pyrite cathode, as already explained, which is then said to be *polarised*. This hydrogen has a solution pressure of its own, and opposes the solution pressure of the gold, which results in a diminution in the current strength as the hydrogen accumulates and becomes more concentrated. The effect of the air is to saturate the solution surrounding the pyrite with oxygen molecules, which induces the occluded hydrogen to become ionised and enter into combination, forming (1) hydrogen peroxide and (2) water thus—



A third reaction occurs between the oxygen and hydrogen ions to form water direct thus—



So long as the flow of air bubbles is kept up, so long will the current remain constant and the gold continue to dissolve at a high and uniform rate. As the area of the gold surface diminishes so also will the current diminish, and in like proportion the weight of metal dissolved in a given time.

The opposing solution pressure of the hydrogen varies with the amount of hydrogen occluded by the pyrite. Thus, when the hydrogen increases in density until its solution pressure is equal to that of the gold, dissolution must cease. In practice, however, this never occurs, for there is always a leakage of current, and consequently metal dissolved, due, probably, to the oxygen in the solution continually combining with some occluded hydrogen. The affinity of hydrogen for oxygen is greater than that of gold for cyanogen, and it follows, if oxygen is available to oxidise the deposited hydrogen, dissolution of the gold can continue; but if no oxygen is available, the opposing forces soon balance each other and dissolution of the metal must cease.

The reason why the air, when applied to the surface of the gold, does not increase the dissolving action, is simply that, at that electrode, it has no function to perform.

Oxidising agents have the same effect, to a greater or less extent, as air



or oxygen gas; but the chemical changes resulting will vary with the oxidiser. They all, however, accomplish the same end, which is to depolarise the negative electrode by combining with hydrogen as deposited.

It is thought by some chemists that base metals and minerals dissolve in the absence of oxygen, but that gold and silver dissolve only in its presence. As a matter of fact, all difficultly soluble metals, and for the most part metallic minerals, only dissolve to an appreciable extent, under ordinary atmospheric changes, in the presence of oxygen or an oxidising agent; the reason being chiefly that their solution pressure is too small to overcome the opposing electromotive force, due to deposited hydrogen at the cathode, whereas the more oxidisable metals produce sufficient energy to overcome this opposing electromotive force, and can therefore proceed to dissolve even in absence of oxygen. The presence of oxygen, however, indirectly makes all metals more soluble.

The action of oxygen may be better understood from the following experiments: Oxygen and hydrogen gases when mixed in a flask may be kept for an infinitely long time without chemically combining, but if a piece of moist platinised platinum be introduced into the mixture, union takes place and water is formed. A similar action occurs if one piece of platinised platinum is charged with H and a second piece with O, and then short-circuited by a wire and immersed in water. A current is set up, flowing from the O electrode through the wire to the H, and thence through the water, the result being a union of the H and O, forming water. When the operation is conducted in presence of air, traces of  $\text{H}_2\text{O}_2$  are also formed, and if air be blown through the solution, the amount of  $\text{H}_2\text{O}_2$  formed is much increased. Again, if the two plates are connected with a primary cell and air is blown through the solution at the cathode, a considerable quantity of  $\text{H}_2\text{O}_2$  may be detected; and finally, if the source of the current be a dissolving metal in the solution and air be blown into the solution at the cathode, the same action takes place, i.e. the formation of  $\text{H}_2\text{O}_2$ .

Now, when  $\text{H}_2\text{O}_2$  is in contact with a cathode charged with H, one atom of the O combines with two atoms of the occluded H, and the result is two molecules of water.

When dissolving gold in a clean cyanide solution in contact with pyrite,  $\text{H}_2\text{O}_2$  can always be detected in the solution, particularly in that part surrounding the pyrite, but the quantity formed varies widely with the impurities in the solution. Bodlaender<sup>1</sup> dissolved gold rapidly in an aerated solution, and not only detected the presence of  $\text{H}_2\text{O}_2$ , but also estimated that the quantity produced was large.

The action may be explained by the H occluded at the electrode exerting its solution pressure, and in the presence of O molecules being induced to break down into ions, two H ions combining with one O molecule. When, however, the conditions admit of O ions also being formed, as in the case above cited, or when a metal capable of occluding both gases is immersed in

<sup>1</sup> *Zeit. angew. Chem.*, p. 583, 1896.

water, and a mixture of oxygen and hydrogen is caused to bubble through the water in contact with the metallic surface, the reaction becomes ionic and water is formed with only traces of  $H_2O_2$ .

The action of the air being entirely local, it cannot interfere with the current between the gold and pyrite electrodes, but as the hydrogen becomes ionised and oxidised, the current increases, by reason of removing the opposing electromotive force. In dry gases no such actions can take place, as the gases cannot become ionised, and therefore cannot combine.

The action of air or oxygen may be retarded, or entirely stopped, by the presence of reducing agents in the solution.

**Effect of Contact with Iron.**—From the electro-chemical point of view it is clear that the best condition for dissolving gold is when it is in contact with a metal which is electro-negative to it in the solvent. If the negative surface is large, the dissolving action will be more rapid, and incidentally oxygen will be unnecessary until the whole negative surface has been polarised. Therefore it follows that a particle of gold in contact with a comparatively large surface of iron will dissolve much more rapidly than when alone in a cyanide solution. This may be proved experimentally in the following way: Place a piece of gold leaf in a cast-iron button mould, with a small iron rod resting on it to press it against the bottom. Put a similar piece of gold leaf in a glass beaker pressed down in the same way by a glass rod; and add cyanide solution to each vessel. In the iron mould the gold dissolves rapidly, in the glass beaker, slowly.

The following quantitative tests, made by C. Toombs at the E.R.P. Mines laboratory prove the accelerating effect of contact with iron:

Three pieces of gold of nearly equal surface area and weight were placed in a 0.25 per cent. cyanide solution, one piece in contact with wrought iron, one in contact with a crystalline fractured surface of cast iron, and one in contact with glass only. After standing for eighteen hours, they were taken out, dried, and weighed, with the following results:

TABLE V.

| Material in Contact. | Weight of Gold taken. | Weight of Gold Undissolved. | Percentage Dissolved. |
|----------------------|-----------------------|-----------------------------|-----------------------|
| Cast iron . . .      | 21.8 mgm.             | 5.0 mgm.                    | 88.5 per cent.        |
| Wrought iron . .     | 21.9 „                | 10.7 „                      | 52.0 „                |
| Glass . . .          | 21.2 „                | 19.6 „                      | 7.5 „                 |

The reason why the cast-iron surface gave a better result than the wrought-iron one is probably because the hydrogen on it escaped more freely from the numerous points of the fractured surface.

## CHAPTER VII.

### DISSOLUTION OF THE GOLD AND SILVER.

#### Section II.

**The Effect of Reducing Agents.**—Potassium cyanide and alkali of commercial quality usually contain sulphides that act as deoxidising agents when in solution. J. Løvvy<sup>1</sup> made some determinations of  $K_2S + Na_2S$  in good commercial cyanide, and found that it contained only 0.37 per cent., and that gold leaf dissolved just as quickly in a 0.5 per cent. solution of this salt as in a similar strength cyanide free from sulphides. He is of opinion, however, that higher percentages of sulphides may alter matters materially. A. F. Crosse<sup>2</sup> says: "The conclusion I am forced to arrive at is that alkaline sulphides are not injurious when present only in very small quantities, if proper regard is had in the use of the cyanide solutions to the aeration of the solution; but a very small quantity of such sulphides may be injurious under conditions unfavourable to aeration of the cyanide solutions." This is certainly true, but if the solution is properly aerated, sulphides can no longer exist.

Sulphides are sometimes introduced into the solution with alkali added to protect the cyanide. Good commercial caustic soda often contains a small percentage of  $Na_2S$  and ferro-sodium sulphide.

Julian and Smart made some investigations on the effect of small quantities of  $Na_2S$  added to a 0.25 per cent. cyanide solution when used for dissolving gold and silver. Table VI. gives the results. The weight of gold dissolved

TABLE VI.

| Na <sub>2</sub> S added per ton<br>Solution.<br>gm. | Relative Dissolving Power. |         |
|---|----------------------------|---------|
|   | Gold.                      | Silver. |
| 0   | 100                        | 55      |
| 5   | 66                         | 42.6    |
| 20  | 57.7                       | 39.6    |
| 40  | 53.4                       | 39      |
| 80  | 52                         | 39      |
| 280   | 50.6                       | 39      |

in a clean solution is taken as 100. That is its dissolving power, and the other figures represent the dissolving powers in terms of that solution.

<sup>1</sup> *Proc. Chem. and Met. Soc. S. Africa*, vol. i. p. 87.

<sup>2</sup> *Ibid.*, vol. i. p. 89.

These results have been plotted in Fig. 17, and the two curves placed beside each other for comparison. It is interesting to note that the solubility of

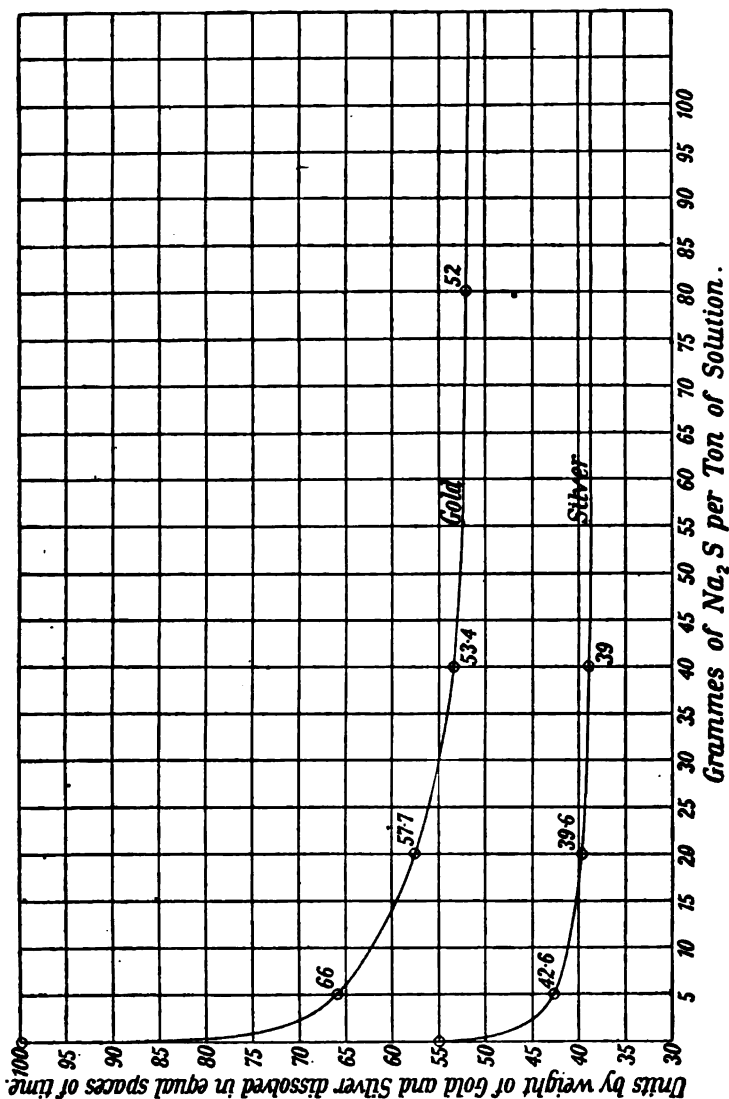


FIG. 17.—The effect of Sodium Sulphide in a 0.25 per cent. KCN Solution on the rate of dissolution of Gold and Silver.

silver is not affected to anything like the same extent as gold. In weaker cyanide solutions the silver becomes more affected and the gold less.

Many cyanide engineers have noticed that when a new solution has passed through an ore containing sulphides, tellurides, and other minerals, it is no

longer capable of dissolving the same quantity of gold in the same time as the original solution, although containing the same percentage of cyanide.

In order to prove that this action is real and not imaginary, and may be traced to some change in the solution, the following experiments were made: A well-mixed sample of concentrates, containing chiefly iron pyrites (pyrite and marcasite), was divided into eight equal parts, and each part was placed in a separate treatment vat. These were then thoroughly washed, each in the same way, and a cyanide solution (0.3 per cent.) equal in weight to the ore, was slowly passed through No. 1 vat with a measured quantity of air. The filtrate was then passed on to No. 2 vat, and so on through all the eight vats, each having been dealt with in identically the same way as No. 1. The results are given in Table VII.

TABLE VII.

Assay of Originals, 92 dwt. Gold.

| No. of Vat. | Cyanide per cent. | Assay of Residues, dwt. | Gold per cent. extracted. |
|-------------|-------------------|-------------------------|---------------------------|
| 1           | 0.3               | 27.6                    | 70                        |
| 2           | 0.3               | 36.55                   | 60.3                      |
| 3           | 0.3               | 45                      | 51.1                      |
| 4           | 0.3               | 51.24                   | 44.3                      |
| 5           | 0.3               | 56.9                    | 38.2                      |
| 6           | 0.3               | 59                      | 35.9                      |
| 7           | 0.3               | 60.7                    | 34                        |
| 8           | 0.3               | 60.6                    | 34.1                      |

The solution had lost cyanide in passing through the vats, but was in each case made up to the original strength, 0.3 per cent., before passing on to the next vat. The solution was applied with a sprayer at a uniform rate, which gave it a better opportunity of taking up oxygen than would be obtained in practice. The time of treatment in all cases was the same, and, as far as practicable, identically the same method of procedure was maintained throughout. The residues were washed with a superabundance of air-free water before assaying. It is noteworthy, that in these experiments, the volume of air passed through the ore with the solution has to be regulated and maintained alike in each case, or the results become erratic. The larger the volume of air used the worse is the resulting filtrate for re-use in dissolving gold and silver, but, as a rule, the better is the extraction of the metal from the ore treated.

It often happens that silver predominates in an ore, and it seemed probable that with such an ore, the above changes in the solution would have even a more marked effect. A mispickel ore, in which silver predominates, behaved identically in the same manner as the case above cited, the solution used containing 0.4 per cent. cyanide. The results are given in Table VIII.

The ore was one that was particularly suitable for treatment by the cyanide process, for by repeated treatments with clean cyanide solution 98·8 per cent. of the silver was extracted and about 80 per cent. of the gold.

That a change takes place in the solution in leaching the ore is beyond doubt, but what that change is requires investigation. After passing through eight vats, the solution in each case was of a dingy brown appearance. This changed to an amber tint after standing in contact with air and direct

TABLE VIII.

Assay of Originals 18·3 dwt. Gold.  
 „ „ 60·1 „ Silver.

| No. of Vat. | Cyanide per cent. | Silver in Residues, dwt. | Per cent. Silver extracted. | Gold in Residues, dwt. | Per cent. Gold extracted. |
|-------------|-------------------|--------------------------|-----------------------------|------------------------|---------------------------|
| 1           | 0·4               | 18                       | 70                          | 6·8                    | 65                        |
| 2           | 0·4               | 25·1                     | 58·2                        | 11·85                  | 35·2                      |
| 3           | 0·4               | 30·36                    | 49·4                        | 12·88                  | 29·6                      |
| 4           | 0·4               | 34·8                     | 42·1                        | 13·5                   | 26·2                      |
| 5           | 0·4               | 38·8                     | 36·3                        | 14·2                   | 22·4                      |
| 6           | 0·4               | 42                       | 30                          | 14·44                  | 21·1                      |
| 7           | 0·4               | 42·6                     | 29·1                        | 14·5                   | 20·9                      |
| 8           | 0·4               | 42                       | 30                          | 14·6                   | 20·3                      |

sunlight for about a week, but on adding an oxidising agent to a portion the change of colour was complete in a few minutes.

The solution in the first case, which had been a week in contact with air and sunlight, was then strengthened to 0·3 per cent. cyanide and passed through a ninth vat of the same ore in exactly the same way as the other eight solutions. The assays showed an extraction of 66·6 per cent. of gold as against 70 per cent. with the new solution. It would therefore appear that the solution had almost recovered its original dissolving power.

In the second case the solution was immediately treated with  $\text{Na}_2\text{O}_2$  and strengthened to 0·4 per cent., and within half an hour was applied to a ninth vat of ore, resulting in an extraction of 65·8 per cent. silver and 60·2 per cent. gold. From these experiments it may be assumed that reducing agents are accountable for the loss in dissolving power of the cyanide solution, and that this power may be regained by the addition of oxygen or oxidising agents.

**Removal of Reducing Agents.**—It is supposed that the reducing agents present which cause loss in dissolving power of the solution are alkaline sulphides, but it is probable that other reducing agents are also accountable for the action. If sodium sulphide be dissolved in water it slowly changes, in contact with air, to caustic soda and thiosulphate of sodium. The necessary oxygen for this change must come from that dissolved in the solution. The oxidising of  $\text{Na}_2\text{S}$  into  $\text{Na}_2\text{S}_2\text{O}_3$  by this means is retarded by

the presence of caustic soda in excess, but hastened in presence of a carbonate or  $\text{CO}_2$ . The same action takes place when an energetic oxidiser, as  $\text{O}_3$  or  $\text{H}_2\text{O}_2$  is employed, but the oxidation is more complete, forming finally  $\text{Na}_2\text{SO}_4$ .

If cyanides are present in the solution a small portion also becomes oxidised by the action of the air, and with more energetic oxidisers the amount may be much increased. With weak solutions the loss from this source is, however, not sufficient to be a matter of importance. A little sulphur combines with the cyanide to form thiocyanate, but this compound does not appear to have any retarding influence on the dissolving of gold and silver.

The oxidising of sulphides introduces, into the solution, compounds that may not have any marked influence on the dissolving power of the cyanide, but the increase of salts in the solution, even inert salts, is generally undesirable. Attempts have been made to remove the sulphur as insoluble compounds, and in March, 1893, J. S. MacArthur and C. J. Ellis patented the idea underlying the addition of a lead salt to the solution, which had for its object the removal of sulphur as insoluble lead sulphide. Other desulphurising agents were also suggested. This method will remove excess of sulphur, but the action, when only traces of sulphide are present, as is usually the case, is not very effective. Even after the most complete precipitation of sulphur, the solution should be thoroughly aerated or treated with an oxidiser before it is used.

**Lead Salts in Practice.**—Lead acetate is generally used as a corrective, but J. E. Clennell<sup>1</sup> finds that almost any lead compound, soluble or insoluble, except the tartrate, will give equally good results, the effect being proportional to the quantity of lead present. He has also shown that there is generally a point of maximum efficiency beyond which any further addition of lead is detrimental to the extraction of silver. This maximum must be found by experiment in each case.

Details of practice in the use of lead salts have been given in a comprehensive article by M. W. von Bernewitz<sup>2</sup> who shows that the amount used in the treatment of Mexican silver ores varies from 0.1 to 3 lb. per ton of ore. The use of lead salts is very general in the treatment of gold ores and varying quantities of nitrate or acetate are used, according to the amount of sulphides present and the practical results following the addition of the corrective salt.

The question has been raised as to the condition in which lead remains after the acetate has been added to the solution. H. Pearce<sup>3</sup> thinks it probable that the acetate is oxidised into the sulphate, which is stable and insoluble; whilst W. A. Caldecott<sup>4</sup> suggests that the lead exists as an alkaline plumbite.

<sup>1</sup> *Eng. and Min. Journ.*, Sept. 28, 1912.

<sup>2</sup> *Min. and Sci. Press*, Nov. 15, 1913.

<sup>3</sup> *Proc. Chem. Met. and Min. Soc. of S.A.*, June 1908.

<sup>4</sup> *Ibid.*, July 1908.

**Potassium Permanganate in Practice.**—Morris Green<sup>1</sup> has shown that, when potassium permanganate is added to KCN solution, the mixture of precipitated oxides of manganese which is formed constitutes the useful and active oxidising agent. In the Rand method of adding the permanganate, these precipitates are filtered out by the top layers of sand.

**Manganese Dioxide in Silver Ores.**—In the case of some silver-gold ores carrying from 10 to 16 per cent. of manganese dioxide it has been found impossible to extract a profitable percentage of the silver. Byron Jackson<sup>2</sup> suggests that this may be due to the oxidation of the cyanide into cyanate and other compounds. W. H. Coghill<sup>3</sup> disproves this in pointing out that the amount of gold dissolved from the same ores is not influenced by the presence of  $MnO_2$ .

**The Application of Oxidisers.**—In 1890–91 Julian made a number of experiments on the use of air blown through working solutions, and on the addition of more energetic oxidising agents, for the purpose of facilitating the dissolution of gold in ores. The results were so promising that in February, 1892, he covered the use of air blown through the solution, and of the use of the more energetic oxidising agents for this purpose, by Transvaal patent No. 335. About the same time, C. Moldenhauer, of Frankfort-on-Main, was experimenting in the same direction, and patented in the United States in the same month and the same year the use of ferricyanide of potassium as an oxidiser in conjunction with potassium cyanide, to hasten the dissolving action on gold and silver. It was supposed at that time that the only use of oxidisers was merely to supply the oxygen direct for the so-called Elsner reaction instead of taking it from the air; but it is now known that the direct use of oxidisers for dissolving purposes has a very limited application, and that their chief value lies in oxidising reducing agents in the solution.

Julian experimented with a large number of organic and inorganic oxidisers, and it was found that many are undesirable, owing to their forming soluble compounds that increase the density of the solutions, and in some cases forming complexes with the cyanide of no value in the process. This increase of density has generally the effect of decreasing the solubility of the metal, by offering greater resistance to the diffusion of the dissolved oxygen molecules. It is therefore advisable, when employing oxidisers, to select one that introduces only oxygen or oxygen water. The more energetic oxidisers, as  $KMnO_4$ ,  $Na_2O_2$ , and  $O_3$ , oxidise cyanide to cyanate, and facilitate the formation of some complex compounds, but in very dilute solutions this action is not marked.

The solution should be charged with oxygen before being applied to the ore, and reducers oxidised as completely as time will admit. This may be done by blowing in air vigorously for a few minutes at intervals of an hour, but the action is slow, and in cold weather it may take some days, in this

<sup>1</sup> *Journ. Chem. Met. and Min. Soc. of S.A.*, Feb. 1914.

<sup>2</sup> *Min. and Sci. Press*, May 25, 1912.

<sup>3</sup> *Ibid.*, June 1, 1912.



way, to render reducers inert, and saturate the solution contained in large tanks. If the operation is kept up continuously a considerable volume of HCN may be carried off, besides being a source of unnecessary expense. Another method is to pump the solution from one vat to another, employing a pump with a leaky suction pipe. Air is then drawn in with the solution and a thorough mixture ensured, especially if the pump is one of high velocity, like those of the centrifugal type. The selection of methods must, however, be largely governed by cost. Both the methods described have the advantage of thoroughly mixing the solution and making it of uniform strength.

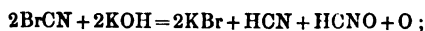
Probably at no works is sufficient time allowed to oxidise completely all reducing agents and re-charge the solution with oxygen by such methods alone, and from Fig. 17 it will be seen how injurious is even a mere trace of sodium sulphide. Where time of treatment is not a matter of importance, the oxidation of sulphides is not so necessary, for if the dissolving power of the solution is reduced by their presence, it is only necessary to increase the time of treatment in the inverse proportion. It is noteworthy that in plants where the sump capacity is small, high residues much more often occur than in works where the capacity is large. This may be accounted for by the solution being longer in contact with the air before it is applied to the ore, thereby preventing the accumulation of reducers.

Oxidisers are sometimes added as solution to the ore before applying the cyanide solution. This is found beneficial when the ore has become partly oxidised, and in the case of slime treatment by agitation. Some of the more easily soluble mineral compounds are then made less soluble in the cyanide solution.

Strong oxidisers, as originally patented by Julian and Moldenhauer, have been applied to the treatment of tailings and concentrates in many parts of the world, and for the most part, with the former material, have not proved economically beneficial. In 1894, Sulman and Teed brought to the notice of engineers the use of a halogen compound of cyanogen, as BrCN, in a cyanide solution, for hastening the dissolution of gold and silver. This compound has been used at several mines, particularly in W. Australia. There has been much discussion as to the action of BrCN, and it has been found that its only action of value is that of an oxidising agent. It has been stated by Sulman and others that it liberates cyanogen. This it probably does, like other oxidising agents, but, as pointed out by W. Skey,<sup>1</sup> "aqueous solutions of cyanogen do not exert the least solvent action on gold and silver." The liberating of cyanogen is really a weak point in the use of strong oxidisers, which will be seen from the reaction—

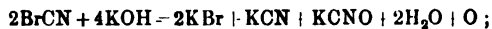


The reactions are similar to those that would occur with other oxidising agents, such as  $\text{K}_3\text{Fe}(\text{CN})_6$ , or  $\text{KMnO}_4$ , and are probably as follows:—



<sup>1</sup> *Eng. and Min. Jour.*, p. 163, 1897.

or with excess of KOH,



with KCN,



or with KCN and an excess of KOH,



The effect of such oxidisers may be demonstrated by the use of apparatus Fig. 16, when it will be found that no action takes place if the oxidiser is applied to the gold surface, but if applied to the pyrite surface a strong current is generated, and consequently a rapid dissolution of the gold must occur.

The value of strong oxidisers when applied to ores is determined largely by the velocity of the reaction. Those which have too high a velocity become wasted because the oxygen is liberated faster than it is wanted to combine with the depositing hydrogen. The liberated oxygen then destroys part of the cyanide. It is obvious that the ideal oxidiser is one by which the oxygen is liberated just at the rate required to combine with the deposited hydrogen. There must, however, always be a large excess of oxygen, as the liberation may not take place at the right point, and reliance must be placed on diffusion.

It is obvious from the above conclusions that it would not be economical to apply strong oxidisers to an ore direct, as, owing to the rapidity of the reaction, the effect lasts too short a time to facilitate the dissolution of the gold to an appreciable extent. When the gold is coarse, a considerable time must be allowed for dissolution, and any oxygen given off beyond that necessary to combine with the hydrogen generated by its dissolution cannot be of service, but may retard dissolution, by destroying cyanide. The only case in which strong oxidisers can be of benefit, when applied to an ore direct, is where the gold is in an extremely fine state of division and can dissolve rapidly, as in the treatment of slime by agitation.

If an oxidising agent is applied to an ore dealt with by percolation, one should be selected that gives off oxygen slowly, as  $\text{MnO}_2$ ,  $\text{BaO}_2$ ,  $\text{PbO}_2$ . These all act as depolarisers, the  $\text{MnO}_2$  being the slowest and the  $\text{PbO}_2$  the most rapid. The peroxides of the alkalies or combinations of  $\text{H}_2\text{O}_2$  are better employed in oxidising reducing agents in the solution before it is applied to the ore.

In some ores the gold is so easily dissolved that it has been said that no marked disadvantage is observed in using solutions of low dissolving efficiency, for the reason, as pointed out by C. Butters and W. Bettel,<sup>1</sup> that it often takes nearly three days to wash out the gold, after it is dissolved, from a vat of ore, and as the washing is done with a cyanide solution, the dissolving of the gold proceeds at the same time. On the other hand, a solution which

<sup>1</sup> *Proc. Chem. and Met. Soc. S. Africa*, vol. i. pp. 331-332.

dissolves the available gold in one day is better than one of lower efficiency, because the earlier the gold is dissolved the better will be the displacement of dissolved metal from the ore.

In practice, moreover, by using clean solutions of high efficiency, uniformly good results are probable, whereas with solutions of low efficiency, the results are usually erratic, even with the simplest class of ore.

When dealing with refractory ores, and especially those containing much silver, it often takes a long time to dissolve the gold, whereas, when the solution is kept up to its highest state of efficiency, the time required may be less than half, and the results more uniformly good.

## CHAPTER VIII.

### DISSOLUTION OF THE GOLD AND SILVER.

#### *Section III.*

**Strong and Weak Solutions.**—The solubility of gold and silver in cyanide solutions is dependent, at ordinary temperatures and atmospheric pressures, on two main factors: (1) the number and velocities of the migratory ions and oxygen molecules that take part in the dissolving of the metals; (2) the difference of potential between two parts of the metallic surface in contact with the solution. Within working limits, the stronger the solution the greater is the number of dissociated molecules or ions in it, but at the same time the more viscous does it become. Increase in viscosity means greater resistance to the movement of the ions and molecules, and therefore the stronger the solution the slower will be their movement. For this reason no proportionality can exist between the strength of the solution and its dissolving power. This will be better understood by reference to Fig. 18. Let the vertical axis on  $OA$  represent equal unit weights of metal dissolved in equal spaces of time, and the horizontal axis on  $OB$  represent the percentage strength of the cyanide solution. Then if the weights of metal dissolved were in proportion to the strength of the solution, the straight line  $OC$  would represent the weights of metal dissolved by the solutions of different strengths, but from a set of experiments the curve  $OD$  was found to represent the weights of metal dissolved. For example, if the weights dissolved were in proportion to the strengths of the solutions,  $b c$ , which is 50 units, would represent the weight of metal dissolved in a 0.5 per cent. solution, but it was found that the actual weight dissolved was only about 25 units, equal to  $b d$ . Therefore  $d c$ , equal to 25 units, shows the decrease in the dissolving power, due chiefly to the increased viscosity of the solution. As might be expected, the more dilute the solution the nearer does the dissolving power approximate to proportionality. Thus it will be seen that for solutions more dilute than 0.1 per cent., the curve  $OD$  nearly follows  $OC$ .

Maclaurin<sup>1</sup> made some interesting experiments to show how the viscosity of a cyanide solution affects the solubility of gold and silver. He found that the amount of metal dissolved in strong solutions was less than the quantity of oxygen absorbed appears to demand, and he thinks that this points to a retarding action by the salt molecules to the motion of the oxygen molecules.

<sup>1</sup> *Jour. Chem. Soc.*, vols. lxvii, lxviii. p. 208.

The potential difference between the gold and the electro-negative surface varies with the latter, although the chemical changes are the same in the same solution. Thus the potential difference between gold and iron pyrite in a cyanide solution is different from the potential difference between gold and galena, although in both cases the gold dissolves, forming  $\text{KAu}(\text{CN})_2$ , the difference being that the one of the greater potential difference dissolves the faster.

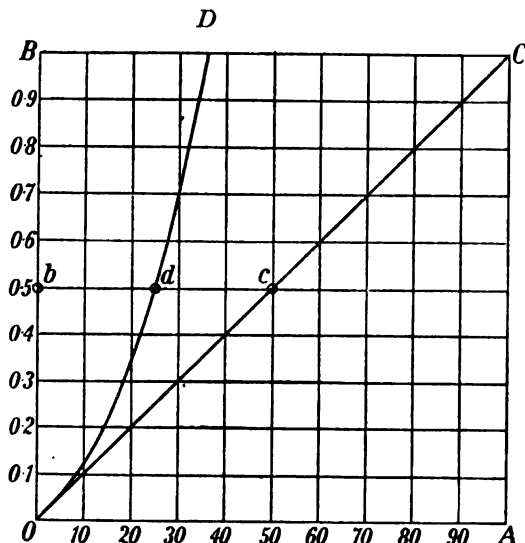


FIG. 18.—Ratio of Weight of Metal dissolved to strength of Solvent.

**Polarisation.**—In the above it has been assumed that there is no polarisation, or, rather, that the hydrogen is oxidised as fast as it is deposited at the negative electrode. In practice, however, polarisation always takes place, and the matter must be considered from a practical point of view. The time of polarisation varies with the relative sizes of the electrodes, that is, with the relative surface areas of the gold and pyrite or other negative substance in contact with the solution, also it varies with the percentage of oxygen absorbed by the solution, and with the resistance offered to the motion of the oxygen molecules in the solution. As it is not possible to determine the values of all these factors, and also as they are constantly varying during the dissolution of the metal, no data are available that represent the loss in solubility of the metal due to polarisation, when treating ores with solutions of different strengths.

What is known, however, is that the stronger the solution the quicker does the electrode polarise; and the greater the negative or pyrite surface, or the smaller the gold surface, the slower does polarisation take place. Also, the stronger the solution the greater is its viscosity; and the more viscous the solution, the greater is the resistance offered to the diffusion of the oxygen molecules through it, therefore the slower is depolarisation.

It is evident there must be a solution of a maximum strength with a minimum resistance to the oxygen molecules. This would have a maximum dissolving power, but the strength must necessarily vary with different sets of conditions. In practice no set of conditions remains constant, but changes during the dissolving of the gold, and changes with ores from different mines, or even the same mine. It is therefore possible only to find a solution that is a rough approximation to one of maximum dissolving power. This can so far be found only in an empirical way, and the composition must obviously vary widely in different localities.

**Physical Effect of Strong and Weak Solutions.**—In practical work the strength of the solution was found, in some cases, to have a marked physical effect on the dissolving of gold and silver. This is most noticeable in ores containing a preponderance of silver over gold in presence of sulphides. Thus it was found that, when employing a weak solution of 0.15 per cent. KCN, a maximum extraction point of 66.6 per cent. silver was reached, and even when the solution was afterwards strengthened to 0.5 per cent. the additional extraction was very small. When, however, the first solution applied was 0.5 per cent., and afterwards weaker solutions were used, the extractions were always over 90 per cent.

It is surmised that the weak solution produced a hard insoluble film of sulphide over the surface of a portion of the metallic silver particles, which prevented even the strong solution from attacking the silver, whereas, when the strong solution was used at first, the sulphide adhered as a loose slimy deposit, which did not prevent the weaker solutions afterwards attacking the metallic silver. A hot solution produces the same slimy deposit, even when the solution is weak.

**Comparative Solubility of Metals and Minerals in Cyanide Solutions.**—J. S. Maclaurin<sup>1</sup> made some interesting investigations on the solubility of gold and silver in strong and weak cyanide solutions. He dealt with plates of pure gold and silver, and found that the rate of dissolution of both metals varies with the strength of the solution, being small for strong solutions, and increasing as the solution becomes weaker until a maximum at 0.25 per cent. KCN is reached, and then again diminishing.

Maclaurin used pure gold plates, in which case the difference of potential between the dissolving part or anode, and the polarisable part or cathode, is at a minimum. Under such circumstances the rate of dissolution of the gold must be small, as its electro-motive force is soon balanced by the opposing electro-motive force of the deposited hydrogen, and all action would cease unless oxygen molecules struck the polarised part and effected oxidation of the hydrogen. In dealing with a pure gold plate in a cyanide solution, Julian and Smart found that the difference of potential, at any two parts of its surface, is  $\frac{1}{200}$  to  $\frac{1}{100}$  of the difference of potential between gold and marcasite. This accounts for pure metal being much less soluble than gold in ores.

In ores, no such conditions as those in Maclaurin's investigation are

<sup>1</sup> *Jour. Chem. Soc.*, vols. lxvii., lxviii. pp. 199-212.

encountered, for a small speck (often microscopic) of gold and silver is usually found in contact with a comparatively large negative area of pyrite or other mineral. In this case there is a much greater difference of potential, which causes the rate of dissolution to be proportionately greater. Complete polarisation never occurs while oxygen is present in the solution, for when the negative area is large compared with that of the gold, it polarises more slowly, and favours depolarisation by meeting with a larger number of oxygen molecules than if the reverse was the case. Also the gold and silver particles are often so small, compared with the pyrite surface, that the metals dissolve completely before polarisation has any appreciable effect. Julian and Smart found that with positive and negative electrodes of the same size, the time of practically complete polarisation between two pure gold electrodes was under one minute, whereas, under the same conditions, where the electrodes were gold and iron pyrites, polarisation was not complete in half an hour.

In dissolving pure gold and silver plates in a cyanide solution, the amount of oxygen dissolved has a sensible effect on the solubility of the metals, but in the case of ores, where large negative and small positive surfaces present themselves to the solution, the effect of oxygen is not so marked, and in fact varies but slightly within comparatively wide limits. The large excess of oxygen is, in the latter case, used up in oxidising impurities dissolved in the solution, such as sulphides. The full benefit of the oxygen for depolarising purposes is, therefore, not obtained.

Much confusion exists in the minds of cyanide men and others as to the meaning of the expression 'comparative solubility of metals.' It may be taken as meaning the comparative weights of metal dissolved from surfaces of equal area in the same time and under the same set of conditions. It is often confounded with the energy with which a metal is attacked by a solution, whereas in reality there is no proportionality between the two. Thus it is often said, because of the energy displayed in dissolving, that aluminium in a cyanide solution is more soluble than zinc, and zinc is more soluble than copper, but from experiment it has been found that, under similar conditions, the weights of these metals dissolved are in the order 1, 3, and 2, respectively, whereas the ratio of the energy produced is about 100, 93, and 84, respectively.

It was thought by some metallurgists that the proportion of gold and mineral matter of the ore dissolved in a weak solution, differed from the proportion dissolved in a strong solution. That is to say, that the ratio of gold to pyrite dissolved in a weak solution differs from the ratio of that in a strong solution. Julian and Smart made some investigations on the subject, and found that this is not the case, but that the same proportionality exists with solutions of all strengths within working limits. The data obtained are given in Table IX. The first vertical column gives the order of solubility, and the other vertical columns give the ratios of solubility in solutions of strengths indicated at the top of each. The horizontal rows of figures indicate the ratios of the weights of metal or mineral dissolved in solutions

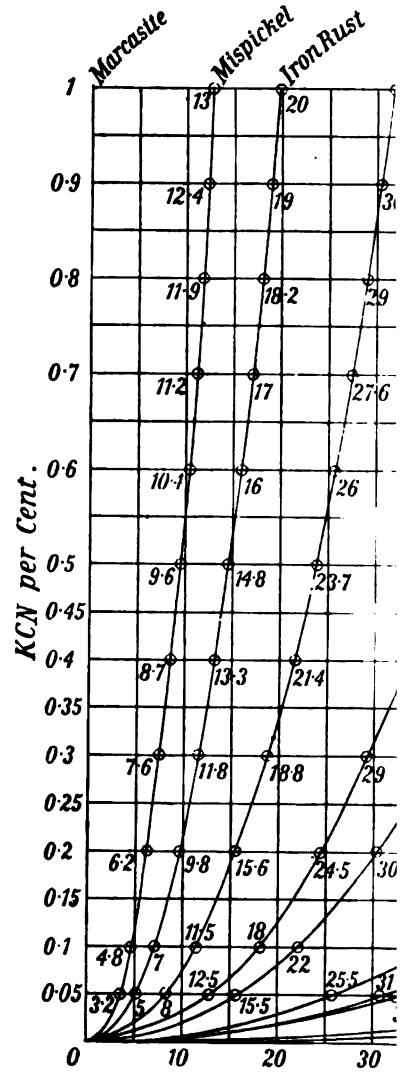
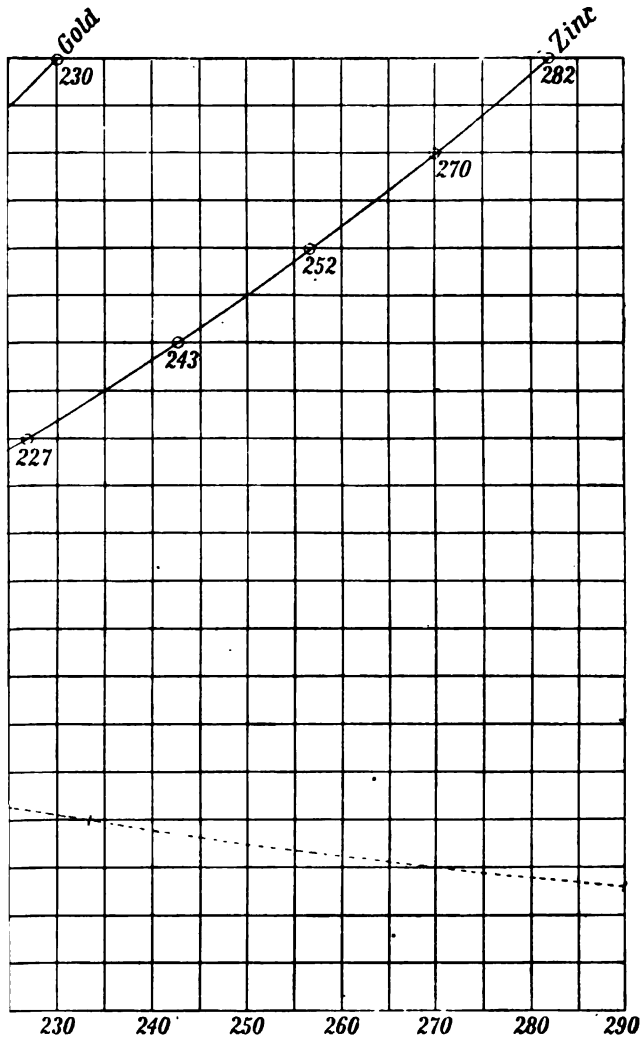


FIG. 11





of KCN per unit of 0.01 per cent.

of the different strengths, from surfaces of equal area in equal spaces of time. These results are shown graphically in Fig. 19.

TABLE IX.

| Order of Solubilities.                                | Per cent. KCN. |       |       |       |       |       |       |       |       |       |       |  |
|---|----------------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|--|
|   | 1              | 0.9   | 0.8   | 0.7   | 0.6   | 0.5   | 0.4   | 0.3   | 0.2   | 0.1   | 0.05  |  |
| Zinc, . . . . .                                       | 232            | 270   | 232   | 243   | 227   | 209   | 190   | 166   | 138   | 100   | 71    |  |
| Gold, . . . . .                                       | 230            | 220   | 210   | 198   | 185   | 170.5 | 154   | 135   | 112.5 | 82.5  | 58    |  |
| Silver, . . . . .                                     | 127            | 121.5 | 116   | 109.5 | 102   | 94    | 85    | 75    | 62    | 45.5  | 32    |  |
| Lead, . . . . .                                       | 121            | 115   | 110.5 | 104   | 97    | 90    | 81    | 71    | 59    | 43    | 31.3  |  |
| Copper pyrites, . . . . .                             | 101            | 98.5  | 92    | 87    | 81    | 75    | 67.5  | 60    | 50    | 36    | 25.5  |  |
| Charcoal iron, . . . . .                              | 62             | 59    | 56    | 53    | 50    | 46    | 41.5  | 36.5  | 30.3  | 22    | 15.5  |  |
| Pyrite and cast iron, . . . . .                       | 50             | 48    | 45.5  | 43    | 40    | 37    | 33.5  | 29    | 24.5  | 18    | 12.5  |  |
| Galena, . . . . .                                     | 52             | 50.5  | 48    | 46    | 43    | 40    | 37    | 33.5  | 29    | 24.5  | 18    |  |
| Iron rust, . . . . .                                  | 20             | 18    | 17.2  | 17    | 16    | 14.8  | 13.3  | 11.8  | 9.8   | 7     | 5     |  |
| Mispickel, . . . . .                                  | 13             | 12.4  | 11.9  | 11.2  | 10.4  | 9.6   | 8.7   | 7.6   | 6.2   | 4.8   | 3.2   |  |
| Marcasite, . . . . .                                  | 0              | 0     | 0     | 0     | 0     | 0     | 0     | 0     | 0     | 0     | 0     |  |
| Efficiency of KCN }<br>per unit.                      | 0.096          | 0.102 | 0.109 | 0.118 | 0.129 | 0.142 | 0.160 | 0.188 | 0.224 | 0.244 | 0.483 |  |
| Relative dissolving }<br>efficiency of solu-<br>tion. | 9.58           | 9.17  | 8.75  | 8.25  | 7.71  | 7.10  | 6.42  | 5.63  | 4.69  | 3.44  | 2.42  |  |

As might be expected, the results obtained with the minerals varied widely with samples from different localities, and therefore an average sample was dealt with. Of the more common metallic minerals, marcasite was found to be the least soluble in cyanide solutions. This is put at zero—an assumption which, of course, is not strictly accurate, but gives a basis for comparison. The solubilities of the other minerals given are all greater than zero, the ratios being indicated in the columns opposite the substance.

The figures were obtained under conditions that produce no polarisation. This is essential for purposes of comparison, as it would be difficult, if not impossible, to maintain the same degree of polarisation during each successive set of determinations. A large number of the values were obtained from actual experiment, and the others were determined by calculation.

TABLE X.

|                   | Per cent. KCN. |         |         |         |         |         |         |         |
|-------------------|----------------|---------|---------|---------|---------|---------|---------|---------|
|                   | 1              | 0.5     | 0.25    | 0.1     | 0.25    | 0.02    | 0.01    | 0.005   |
| Gold, . . . . .   | 0.00650        | 0.00670 | 0.00684 | 0.00675 | 0.00666 | 0.00613 | 0.00345 | 0.00030 |
| Silver, . . . . . | 0.00395        | 0.00400 | 0.00410 | 0.00396 | 0.00380 | 0.00285 | 0.00213 | 0.00040 |

Maclaurin<sup>1</sup> made a number of determinations of the solubility of gold and silver in solutions containing percentages of cyanide similar to the above. Table X. contains his figures, arranged for comparison.

<sup>1</sup> *Jour. Chem. Soc.*, vols. lxvii., lxviii. pp. 199-212.

By Maclaurin's method of making these determinations it would have been impossible to prevent polarisation, although he took the precaution of giving motion to the plates, and of keeping the solution charged with oxygen. In dealing with pure gold or silver plates in a cyanide solution, without being in contact with anything more electro-negative, the difference of potential between any two parts of their surface is so small that an infinitesimal amount of hydrogen deposited on a part of its surface reduces the solubility in a marked degree. It would be expected that this deposition of hydrogen would have less effect in a weak solution, as the oxygen meets with less resistance, and therefore diffuses more readily to the part where the

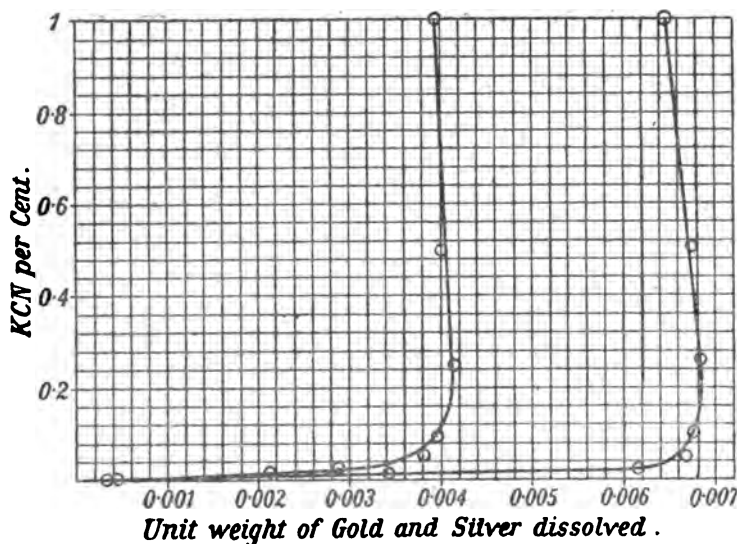


FIG. 20.—Maclaurin's Curves, showing Solubility of Gold and Silver plates in solutions ranging from 0 to 1 per cent. KCN.

hydrogen is deposited, and effects depolarisation. The solubility in the weak solution should therefore approximate to that when no polarisation occurs, but as the solution becomes stronger the resistance to the movement of the oxygen molecules increases, and consequently the solubility of the metal decreases. This is really what occurs, as shown by Maclaurin's results, which are reproduced graphically in Fig. 20.<sup>1</sup> (See Table X.)

It is not possible to place these curves beside the curves in Fig. 19 for want of data to determine the scale, but they are given to show the general direction the solubility of the metals takes in an extreme case, that is, when the gold and silver occur in ores without being in electrical contact with a more negative substance. When the gold and silver are in very fine states of division and in electrical contact with a comparatively large negative area, as pyrite, the general direction of solubility approximates to that given in Fig. 19.

Julian and Smart made a number of exact determinations of the solubility

<sup>1</sup> *Jour. Chem. Soc.*, vols. lxvii., lxviii. p. 206.



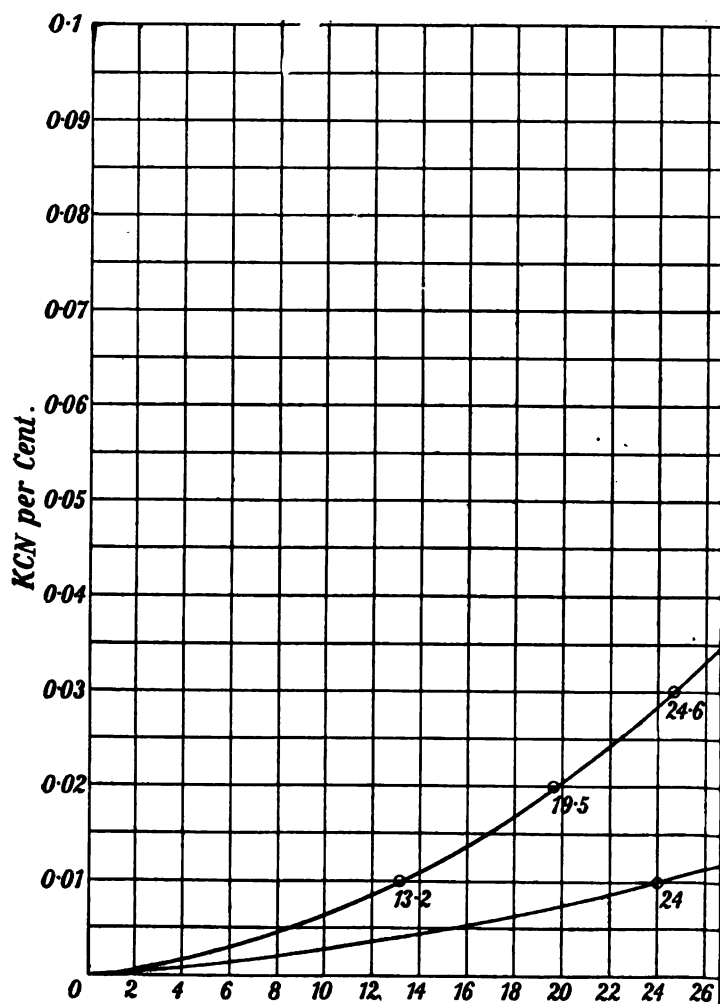


FIG. 21.—The solubility of Gold and Silver,—showing the rat

of gold and silver in very dilute solutions, in a similar manner to those given in Table IX. The results are given in Table XI., and are shown graphically in Fig. 21 on an enlarged scale.

**Efficiency of Strong and Weak Solutions.**—On studying Tables IX. and XI. it will be seen that there is no proportionality between the weight of metal dissolved and the percentage strength of the solution. Thus a 0·01 per cent. solution dissolves 24 of gold, whereas a 0·1 per cent. solution, ten times as strong, dissolves only 82·5 in the same time, and a 1 per cent., or one hundred times as strong, dissolves only 230 parts instead of 2400, as would be the case if proportional.

Therefore the weaker the solution the greater is its efficiency per molecule of KCN dissolved, but in practical work this rule can easily be carried too far. A solution that would have a maximum dissolving power and a maximum efficiency per molecule of KCN is what is desired for working strength. This can, however, be arrived at only in an empirical way, as the time factor has to be taken into account.<sup>1</sup> Let 0·01 per cent. be assumed as a minimum working strength, called the *working unit*; it is then possible to obtain

TABLE XI.

| Order of Solubilities.                       | Per cent. KCN. |       |       |       |       |       |       |       |       |      |
|--|----------------|-------|-------|-------|-------|-------|-------|-------|-------|------|
|  | 0·1            | 0·09  | 0·08  | 0·07  | 0·06  | 0·05  | 0·04  | 0·03  | 0·02  | 0·01 |
| Gold, . . .                                  | 82·5           | 78·5  | 74    | 69·25 | 64    | 58    | 51·5  | 44·5  | 35·5  | 24   |
| Silver, . . .                                | 45·5           | 43·3  | 40·8  | 38·2  | 35·3  | 32    | 28·4  | 24·6  | 19·5  | 13·2 |
| Efficiency of KCN per unit }                 | 0·344          | 0·363 | 0·385 | 0·412 | 0·444 | 0·483 | 0·536 | 0·618 | 0·740 | 1    |
| Relative dissolving efficiency of Solution } | 3·44           | 3·27  | 3·08  | 2·88  | 2·67  | 2·42  | 2·15  | 1·85  | 1·48  | 1    |

the dissolving power per working unit of KCN, by dividing the given ratios of the weights of metal dissolved by the number of working units in the strength of the solution. Thus, in the case of gold, Table XI., the weight of metal dissolved in a solution when the strength is unity (0·01) is 24, so that the dissolving efficiency of this solution is 24. In a solution of 0·1 per cent., which contains ten working units:  $82·5 \div 10 = 8·25$  per unit; and in a 1 per cent. solution, which has one hundred working units: 2·3 per unit. That is to say, the dissolving efficiencies of 0·01, 0·1, and 1 per cent. solutions are 24, 8·25, and 2·3 per unit of KCN, respectively.

<sup>1</sup> If the time factor is left out of the question, as in cases where the gold and silver dissolve rapidly, but can be washed out of the ore only slowly, the most efficient strength of solution is then from 0·07 to 0·09 per cent. KCN.

The dissolving efficiency of KCN per unit for strengths of solution between 0.01 and 1 per cent. in terms of the working unit (namely the 0.01 per cent. solution which is assumed to have a dissolving efficiency of 1) has been calculated, and for convenience are placed in Tables IX. and XI. The results, multiplied by 1000, are plotted in Fig. 19, and by 100 in Fig. 21.

These efficiencies per unit of KCN are calculated on the figures given for gold, but they are practically identical for all the minerals in the list, except marcasite.

The time required is affected, not by the efficiency per unit, but by the relative dissolving powers of the various strengths of solution. Julian and Smart calculated these on the basis of making the dissolving power of the 0.01 solution equal to unity, the figures being given in the lowest lines of Tables IX. and XI. as the relative dissolving efficiency of the solutions. They are, of course, proportional to the experimental figures in the body of the Table.

If the time required to treat an ore with a solution of a certain strength is known, and if it is desired to vary the capacity of the plant by lengthening or shortening the treatment, one can determine what strengths of solution should give equally good extractions.

If  $T$  be the time of treatment with a given solution of a dissolving efficiency  $d$ ;  $T_1$  the time to be allowed in the altered conditions; and  $d_1$  the dissolving efficiency of the required solution, then :

$$d_1 = \frac{Td}{T_1}$$

Thus, if it required 144 hours to treat a charge of slime with 0.02 per cent. solution, and it becomes necessary to increase the capacity of the vats by shortening the time to 48 hours, then :

$$d_1 = \frac{144 \times 1.48}{48} = 4.44,$$

which corresponds to a solution of 0.2 per cent. It follows, therefore, that in order to reduce the time to one-third it is necessary to use a solution nearly ten times as strong.

## CHAPTER IX.

### TEMPERATURE EFFECTS IN CYANIDING.

**Temperature and Viscosity.**—The term ‘viscosity,’ as applied to liquids, may be defined as ‘internal friction.’ This internal friction varies with the temperature. Thus when water is heated it expands and its viscosity becomes less. Both the settlement of slime and the classification of ore pulp are dependent on the fact that the ore particles are not impeded by the viscosity of the fluid to an extent negating the action of gravity. Hence a reduction in the viscosity of the water, caused by heating the pulp, will materially quicken settlement and assist classification. Decreased viscosity also assists the various chemical reactions taking place in cyanide solutions.

The viscosity of a liquid may be measured in several ways and may be recorded as a coefficient. This index has been defined as “equal numerically to the force necessary to maintain a face of one layer of one unit area past another of the same area with a relative velocity of one unit, the distance between the layers being unity, and the space between them being continuously filled with the viscous substance.”<sup>1</sup>

The difference in the internal friction of water which may be accomplished by an alteration in temperature is shown in the following table :<sup>2</sup>

TABLE XII.

| Temperature,<br>degrees C. | Viscosity,<br>c.g.s. units. | Temperature,<br>degrees C. | Viscosity,<br>c.g.s. units. |
|----------------------------|-----------------------------|----------------------------|-----------------------------|
| 0                          | ·01793                      | 40                         | ·00657                      |
| 10                         | ·01311                      | 60                         | ·00469                      |
| 20                         | ·01006                      | 80                         | ·00356                      |
| 30                         | ·00800                      | 100                        | ·00284                      |

**The Influence of Temperature on Dissolution of Metals.**—H. C. Jones and J. M. Douglas<sup>3</sup> have proved that the amount of ionisation in solutions of salts is not affected by temperature. It seems beyond doubt that the solubility of a metal is dependent largely on the amount of ionisation of the salt or acid, and on the velocity of the ions. The rate of dissolution of a metal is increased with rise of temperature, and as this cannot be due to greater ionisation, it

<sup>1</sup> *Principles of Physics*, by A. Daniels, p. 307.

<sup>2</sup> *Am. Chem. Jour.*, vol. xxvi. No. 5, Nov. 1901.

<sup>3</sup> Hosking, *Phil. Mag.*, 1909.



may be assumed that the increase in dissolution is due largely to an increase in the velocity with which the ions move. There are, however, other reasons for the increase of dissolution.

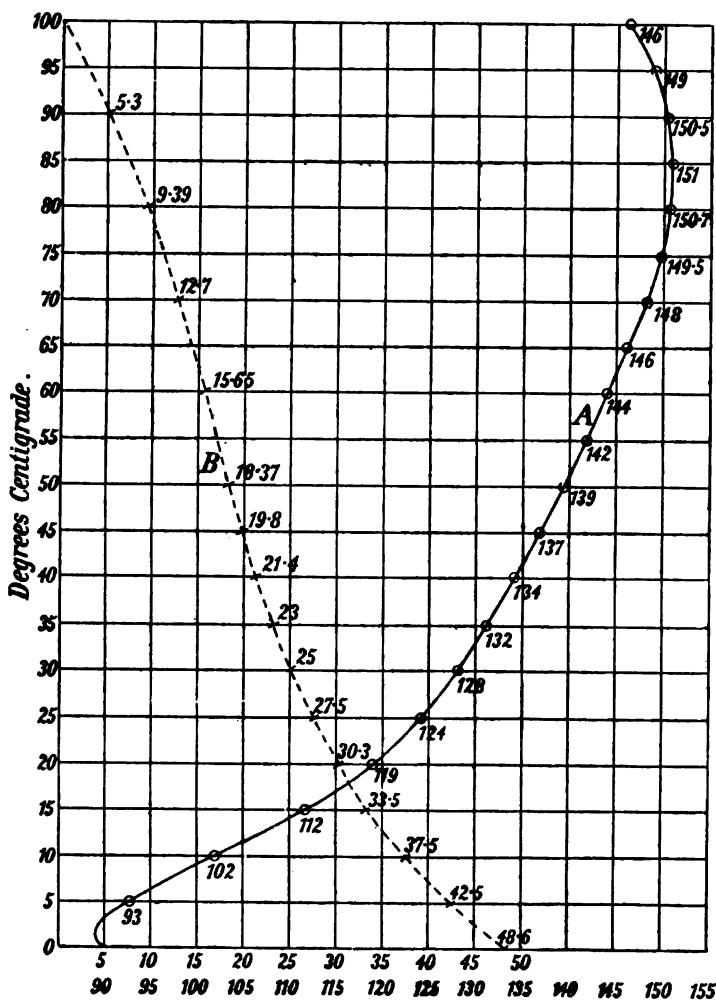


FIG. 22.—Temperature effects,—showing the Solubility of Gold in a 0.25 per cent. KCN solution at temperatures ranging from freezing to boiling points of water, and the relative Solubility of Oxygen in water at the same temperatures.

Julian and Smart made a number of determinations of the solubility of gold in a 0.25 per cent. KCN solution, at temperatures between those of the freezing and boiling points of water. The results are given in Table XIII. and are plotted in Fig. 22. The first and second columns give the temperatures in degrees Centigrade and Fahrenheit. The third column gives the unit weights of gold dissolved in equal spaces of time when no polarisation

occurs. It will be noticed that at 0° C., 90 units were dissolved, and that with increase of temperature the solubility diminishes to 89 at 1 to 2 degrees, and then there is a gradual increase in the rate of dissolution until a temperature of 85° C. (185° F.) is reached, when the rate becomes 151. At higher temperatures there is a gradual decrease, until at boiling point it falls to 146.

TABLE XIII.

| Temperature. |      | Unit Weights<br>of Gold<br>dissolved. | Oxygen<br>Solubility in<br>Water $\times 1000$ . |
|--------------|------|---------------------------------------|--|
| C°.          | F°.  |                                       |  |
| 0            | 32   | 90                                    | 48.6   |
| 1            | 33.8 | 89                                    | 46   |
| 3            | 37.4 | 90                                    | 44.8   |
| 5            | 41   | 93                                    | 42.5   |
| 10           | 50   | 102                                   | 37.5   |
| 15           | 59   | 112                                   | 33.5   |
| 20           | 68   | 119                                   | 30.3   |
| 25           | 77   | 124                                   | 27.5   |
| 30           | 86   | 128                                   | 25   |
| 35           | 95   | 132                                   | 23   |
| 40           | 104  | 134                                   | 21.4   |
| 45           | 113  | 137                                   | 19.81  |
| 50           | 122  | 139                                   | 18.37  |
| 55           | 131  | 142                                   | ...  |
| 60           | 140  | 144                                   | 15.65  |
| 65           | 149  | 146                                   | ...  |
| 70           | 158  | 148                                   | 12.7   |
| 75           | 167  | 149.5                                 | ...  |
| 80           | 176  | 150.7                                 | 9.39   |
| 85           | 185  | 151                                   | ...  |
| 90           | 194  | 150.5                                 | 5.3  |
| 95           | 203  | 149                                   | ...  |
| 100          | 212  | 146                                   | 0  |

With rise of temperature there must be a decrease in the quantity of oxygen absorbed by the solution; nevertheless there is no corresponding diminution in the rate of dissolution, but, on the contrary, the rate steadily increases. No satisfactory data as to the quantities of oxygen dissolved in a cyanide solution at different temperatures were obtainable, but in absence of these, and to meet the purpose, the data given by Winkler<sup>1</sup> of oxygen dissolved in water have been taken, and it has been assumed that in a cyanide solution the amount dissolved would not differ greatly from these. Winkler's results have been multiplied by 1000 and the products placed opposite the different temperatures and plotted as in Fig. 21. By this means the rate of dissolution of the gold may be compared with the relative amount of oxygen absorbed at the various temperatures. It will be observed on examining the table or the curves that the solubility of oxygen decreases with rise of temperature, whereas the solubility of gold increases to a maximum at 85° C. and then slightly decreases to the boiling point, although

<sup>1</sup> *Berichte d. d. chem. Gesell.*, xxiv. 3809.

the oxygen solubility at that temperature is zero. It would appear from this that oxygen has no direct influence on the solubility of the gold. This is what has already been shown to be the case.

In seeking for the reason why, in practice, oxygen should not exert its influence at the higher temperatures as it does at the lower, it has been found that the negative or pyrite electrode polarises to a lesser degree as the temperature rises. That is to say, the capacity of the electrode to adsorb or retain hydrogen at its surface is less in a heated solution than in the cold. This being the case, it follows that the maximum opposing electro-motive force due to polarisation becomes less and less as the solution becomes heated, until the e.m.f. of the dissolving gold overbalances this counter-force, and dissolution of the gold can then go on without the presence of oxygen. True, the e.m.f. of the gold also becomes less as the temperature rises, but this effect is slight compared with the falling off in the opposing e.m.f. due to polarisation.

At the ordinary or low temperatures the balance of the two opposing forces is in favour of the hydrogen, and the deposit of this goes on increasing until its e.m.f. is in equilibrium with the e.m.f. of the gold, and then, in order that dissolution should continue, the presence of oxygen becomes necessary to oxidise the hydrogen. When the electrodes are in this state, for each atom of hydrogen oxidised an atom of gold is dissolved. If there is no oxygen present no hydrogen becomes oxidised, and the two e.m.f.s being in equilibrium, dissolution of the gold cannot proceed until the hydrogen electrode (pyrite) becomes of a lower potential. This may be done in two ways, either by continuously oxidising the hydrogen as it separates, or by raising the temperature to such a point that the maximum e.m.f. of the hydrogen is less than the e.m.f. of the gold.

Referring again to Table XIII. and Fig. 22, it may be noticed how the rate of dissolution of the gold increases with rise of temperature from about 3° or 4° C., at first rapidly, and then more gradually until a maximum is reached, and finally decreases. This maximum point was found for a 0.25 per cent. solution to be at 85° C. (185° F.), but it varies with solutions of other strengths and with impurities in the solution. Decomposition of KCN occurs even at ordinary temperatures, and the rate of decomposition goes on increasing with rise of temperature, but as the temperature rises, the velocities of the K and CN ions increase, as already stated. Therefore, with increase of temperature an increase occurs in the rate of dissolution of the gold, due to increase in the velocities of the K and CN ions, but as the temperature rises a decomposition factor is introduced which reduces the number of K and CN ions, and therefore causes a decrease in the rate of dissolution. Now, it may be conceived that the effect of the reduction in the number of these ions may more than counterbalance the effect of the increased velocity of those remaining, and thereby the net result must be a slower dissolution of the gold. This is evidently why a maximum is reached in the rate of dissolution before the temperature reaches boiling

point. With a weaker solution this maximum is not reached so soon, but with stronger solutions it occurs at lower temperatures, and this is what may be expected, as decomposition and consequent reduction in the number of K and CN ions takes place with weaker solutions at a slower rate.

**Influence of Temperature in Practice.**—It would appear from the foregoing that in practice, by using solutions at high temperatures, there would be a great saving in time or an increase in the extraction. Ores usually contain, besides their gold and silver, one or more metallic minerals, such as iron or copper pyrites or galena, which are also soluble in cyanide solutions, as shown in Table IX. These become more soluble at higher temperatures, just as gold and silver do. Now, at the lower temperatures a portion of the minerals becomes polarised through the dissolving of the gold and silver and other causes, and that part remains, for the time being, more or less insoluble, while at higher temperatures the effect of polarisation is reduced, and the dissolving area of the minerals is increased. A larger quantity of mineral is dissolved in the same time, but the gold and silver, being positive, are not polarised, and so their dissolving area is unaltered by rise of temperature. The net result is a greater proportional dissolution of the mineral matter than the gold and silver, and the consequent reduction in the K and CN ions available for the dissolution of the metals. It is evident, then, that unless the increase in the rate of dissolution of the gold and silver, due to increase of temperature, is more than sufficient to make up for the decrease due to loss of available cyanide, there can be no benefit in using hot solutions.

Again, in practice, when extracting with hot solutions, it is usual to apply the heated solution to the cold ore. In heating the solution it is deprived of its oxygen, as shown in Table XIII., and on being applied to the ore it is cooled. It has been pointed out that at the higher temperatures the presence of oxygen is of little importance, whereas at ordinary or low temperatures it is necessary for the dissolution of the gold or silver. Therefore the cooled solution is not in a fit state for the most efficient extractions.

It has, however, been found possible in practice to overcome these difficulties and to keep the solution at a constant temperature. At some plants the solution is circulated through a heater, generally of the multi-tubular type, placed between the mill engine exhaust and the condenser. In other cases, exhaust steam is delivered into a small tank through which the solution circulated on its way to the treatment plant. In some plants live steam has been used for directly heating the pulp in the agitation vats.

**Summary of Heating Effects.**—Starting from freezing point there is for the first few degrees not only a loss of oxygen but also an increase in density accompanied by increased viscosity. Therefore the first effect would be an actual decrease in the rate of solution of the gold.

As the viscosity becomes less with further heating, and the opposing e.m.f. of the cathode also diminishes, there is a rapid increase in the rate of solution which continues until the decomposition of the cyanide becomes of importance ;

and finally this latter action overcomes the effect of the favourable factors, and the rate of solution again decreases.

In Fig. 22 there is a distinct change of direction in the curve at 25° C. (77° Fah.), so it appears that with the 0.25 per cent. solution used in the experiment the decomposition of the cyanide was already considerable. For a weaker solution, the decomposition would probably only become serious at a higher temperature.

**Practical Application.**—At the Buffalo mill,<sup>1</sup> Cobalt, the 0.3 per cent. solution in the Pachucas is kept at 70° Fah., or over, by live steam in a coiled pipe. At the Tonopah Mining Company's mill the economical limit of heating in the leaching department is stated<sup>2</sup> at 70° Fah. for solutions varying from 0.35 per cent. to 0.25 per cent.

At the Knight's Deep plant, on the Rand, the average temperature used in 1910 in the slimes plant was about 85° Fah., and A. Salkinson<sup>3</sup> considered that more heat would do quicker work. The solutions in this case carried about 0.03 per cent. KCN.

At the Montana-Tonopah mill<sup>4</sup> the pulp in Hendryx agitators is kept at 110° Fah., "by which means the extraction of silver is said to be materially increased." The solution in this case is 0.25 per cent. KCN and the cyanide consumption 3 lb. per ton.

Lengthy investigations were made in the case of ore milled in cyanide solution, at a Rhodesian plant. The observations extended over several months with all practical variations in temperature, and no appreciable difference in the extraction or solution of gold was noticeable. This result is in accordance with the experience of H. A. Megraw who states<sup>5</sup> that "heating solutions seems to have no effect whatever on the extraction of gold." It follows, therefore, that the beneficial results which might be expected from a rise in temperature are, in some instances, counterbalanced by the introduction of unfavourable factors.

**Effect of Temperature on Settlement.**—L. D. Bishop obtained the following experimental results with pulp containing two parts by weight of solution to one of dry slime.<sup>6</sup>

| Temperature,<br>degrees F. | Percentage Volume<br>cleared in 1 hour of<br>Settlement. |
|----------------------------|--|
| 70°                        | 18   |
| 90°                        | 24   |
| 120°                       | 30   |

<sup>1</sup> W. J. Dobbins and H. G. S. Anderson, *Eng. and Min. Jour.*, Aug. 3, 1912.

<sup>2</sup> Claude T. Rice, *ibid.*, June 17, 1911.

<sup>3</sup> *Jour. Chem. Met. and Min. Soc. of S. Africa*, 1909, p. 309.

<sup>4</sup> H. A. Megraw, *Details of Cyanide Practice*, p. 111.

<sup>5</sup> *Details of Cyanide Practice*, p. 135.

<sup>6</sup> *Eng. and Min. Jour.*, April 24, 1909.

These figures, as pointed out by C. W. Comstock,<sup>1</sup> are nearly inversely proportional to the coefficients of viscosity.

The following experimental results of the effect of temperature on the settlement of very fine kaolin were published by M. J. Thoulet, in 1891.<sup>2</sup>

| Temperature,<br>degrees C. | Observed Fall<br>in 80 hours. | Fall averaged<br>from plotted<br>Curve. | Difference. |
|----------------------------|-------------------------------|---|-------------|
| 52                         | 24.25 mm.                     | 23.50                                   | -0.75       |
| 55.5                       | 25.00 "                       | 26.25                                   | -1.25       |
| 56.5                       | 27.00 "                       | 27.00                                   | 0.00        |
| 62                         | 33.00 "                       | 33.00                                   | -1.35       |
| 66.5                       | 34.00 "                       | 34.00                                   | -1.25       |

The plotted curve referred to is given in the original paper and is a straight line which cuts the horizontal axis at 23°, and thereby suggests that, at that temperature, the clay would remain indefinitely in suspension.

In the decantation vats at the Simmer & Jack East slime plant, on the Rand, the average time required for each settlement was reduced from 45 to 29 hours by raising the temperature of the solution to 86.5° Fah., with the result that six vats did more work than nine had done previously with cold solutions.<sup>3</sup>

The beneficial effects of raising the temperature of pulp to hasten settlement or assist classification have not been taken advantage of to any great extent in practice, probably on account of the expense in those cases where exhaust steam is not available. Where an existing plant is overloaded, the method might well be adopted as an economical alternative to plant extension.

**Effect on Percolation.**—A rise in temperature has a marked effect in accelerating percolation, because the chief resistance offered to a solution in passing through a bed of sand is caused by internal friction and, as stated above, this is reduced by heating. If the internal friction at freezing point is 100, at boiling point it would be between 15 and 16.

**Effect on Filtration.**—This has been investigated experimentally by L. D. Bishop<sup>4</sup> with the following result:

| Temperature,<br>degrees Fah. | Lb. of Solution per<br>hour per square foot<br>of Canvas. |
|------------------------------|---|
| 70°                          | 8.25  |
| 90°                          | 18.00   |
| 110°                         | 24.75   |

The filtration was effected by gravity in a Hendryx filter cell.

<sup>1</sup> *Eng. and Min. Jour.*, April 24, 1909.

<sup>2</sup> *Annales des Mines*, vol. xix.

<sup>3</sup> *Jour. Chem. Met. and Min. Soc. of S. Africa*, March 1909.

<sup>4</sup> *Eng. and Min. Jour.*, April 24, 1909.

**Effect on Precipitation.**—The general result of heating solution with respect to precipitation results is seen in an acceleration of the various reactions, beneficial and otherwise, including a more rapid and complete precipitation of the gold and silver. This effect is more marked with very weak solutions than with strong ones.

L. Ehrman<sup>1</sup> stated that with zinc or zinc-copper couple he could precipitate as much gold in two hours at 35° C. (185° Fah.) as in twenty-four hours at 20° C. (67° Fah.).

F. D. Phillips<sup>2</sup> has noted that the amount of white precipitate in the zinc boxes is considerably reduced by heating the solution.

<sup>1</sup> *Proc. Chem. Met. and Min. Soc. of S. Africa*, 1897.

<sup>2</sup> *Ibid.*, 1910.

## CHAPTER X.

### DISSOLUTION OF GOLD PHYSICALLY CONSIDERED.

It may be assumed that, when gold or silver is dissolved in a cyanide solution kept uniform in strength, layers of equal thickness will be dissolved in equal spaces of time. In an ore, the variety of sizes and shapes of the particles of gold must be very great, but a consideration of two or three assumed cases will assist in understanding what actually takes place in practical operations.

(a) The extreme case is when the gold is in thin flat plates, so that its surface is constant, or nearly so, until dissolution is complete. It is evident that in this case, under uniform conditions of the solvent, the quantity dissolved will be in direct proportion to the time, and this may be represented by the straight line *A* in Fig. 23. Again, when gold is embedded in pyrite so that only one edge or surface can be attacked, it would also have a constant rate of dissolution.

(b) The other extreme case is when each gold particle is assumed to be a perfect sphere. Then in each equal interval of time a spherical shell of equal thickness would be dissolved, but each successive shell would be of smaller diameter, and would consequently contain a smaller quantity of gold. Hence in this case the rate of dissolution would be much greater at the beginning than at the end of the operation. Curve *B* illustrates the action for one sphere of any size, or for any number of spheres of equal size.

If, however, in order to approximate more closely to actual conditions, a mixture of spheres of various sizes is assumed, it is clear that the smaller spheres will be entirely dissolved before the larger ones, and in such cases the rate at the beginning will be still more increased.

(c) The curve *C* has been calculated on the assumption that for each three spheres of diameter 1, there are two of diameter 2, and one of diameter 3.

(d) In curve *D* there are one hundred spheres of diameter 1, ten of diameter 2, and one of diameter 3.

The conditions as to relative sizes of particles in actual ores would probably lie between these last two theoretical cases. But as the actual particles of gold are not true spheres, in practice, therefore, the curve would be less convex than *C* or *D*, especially towards the end, because, with any shape except the sphere, the particles would gradually tend towards flatness as they become smaller.



These calculated curves can now be compared with curve *E*, which is plotted from figures given by H. K. Picard<sup>1</sup> of a case at Deloro where one solution was continuously circulated through the ore and cyanogen bromide

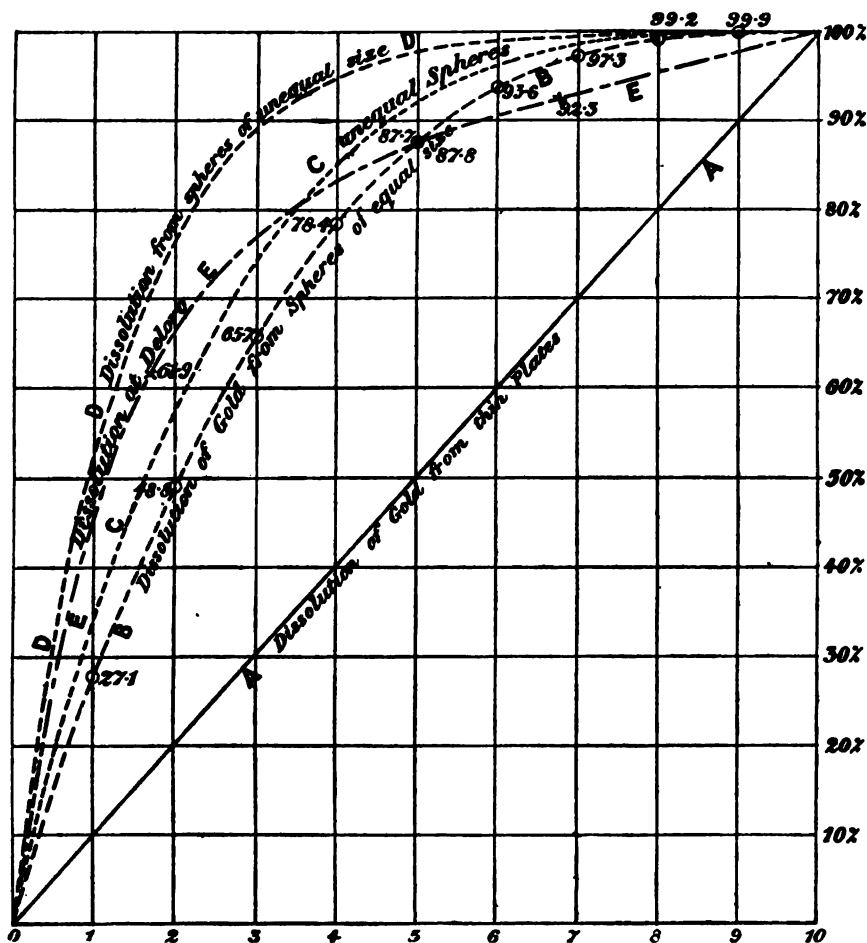


FIG. 23.—Curve *A* shows the rate of dissolution of thin gold plates where the surface exposed to the solution is practically constant; Curve *B*, where the gold is in the form of spheres of equal diameter; Curve *C*, where the gold is in spheres of unequal size: 1 at 3 units, 2 at 2 units, and 3 at 1 unit diameter; Curve *D*, where the gold is in spheres of unequal size: 1 at 3 units, 10 at 2 units, and 100 at 1 unit diameter. Curve *E* is from Picard's figures obtained at Deloro.

added at intervals, so that the conditions of solution must have been fairly constant. The solution was sampled and assayed at intervals. The total gold in the vat was 33.9 oz., the amount dissolved was 28.6 oz., and the percentages have been calculated on the latter figure, because only the gold

<sup>1</sup> *Trans. Fed. Inst. of Min. Eng.*, vol. xv. p. 417, 1897-8.

capable of being dissolved is now being considered. It will be seen that the curve follows very closely the form to be expected from the above reasoning. Up to an extraction of 80 per cent. it lies between *C* and *D*, and from about 88 per cent. it is almost a straight line.

The figures relating to these curves are given in Table XIV., in which the time is divided into ten equal parts. The top row of figures opposite each letter shows the percentage of the total gold which has been dissolved during each interval, and the lower figure gives the total percentage dissolved at the end of each period.

This matter has been treated at some length, because it enables a clear mental picture to be formed of the action taking place in a cyanide vat, and leads to one definite reason why in practice the whole of the soluble gold cannot

TABLE XIV.

|   | 1            | 2            | 3            | 4            | 5           | 6           | 7           | 8           | 9             | 10          |
|---|--------------|--------------|--------------|--------------|-------------|-------------|-------------|-------------|---------------|-------------|
| A | 10%<br>10    | 10<br>20     | 10<br>30     | 10<br>40     | 10<br>50    | 10<br>60    | 10<br>70    | 10<br>80    | 10<br>90      | 10<br>100   |
| B | 27.1<br>27.1 | 21.7<br>48.8 | 16.9<br>65.7 | 12.7<br>78.4 | 9.3<br>87.7 | 5.9<br>93.6 | 3.7<br>97.3 | 1.9<br>99.2 | 0.7<br>99.9   | 0.1<br>100  |
| C | 33.6<br>33.6 | 24.0<br>57.6 | 16.5<br>74.1 | 11.0<br>85.1 | 7.0<br>92.1 | 4.1<br>96.2 | 2.2<br>98.4 | 1.1<br>99.5 | 0.4<br>99.9   | 0.1<br>100  |
| D | 50.2<br>50.2 | 26.8<br>77.0 | 12.1<br>89.1 | 5.7<br>94.8  | 3.0<br>97.8 | 1.4<br>99.2 | 0.5<br>99.7 | 0.2<br>99.9 | 0.09<br>99.99 | 0.01<br>100 |
| E | 45.2<br>45.2 | 20.9<br>66.1 | 10.7<br>76.8 | 6.4<br>83.2  | 4.6<br>87.8 | 2.9<br>90.7 | 2.4<br>93.1 | 2.4<br>95.5 | 2.8<br>97.8   | 2.2<br>100  |

*Note.*—Top figure in each square = % dissolved during interval. Lower figure = total % dissolved at end of each interval.

be dissolved. It will be noticed that, except in case *A* (which is purely ideal), more than 90 per cent. of the gold is dissolved in six-tenths of the time required for complete dissolution, and over 95 per cent. in eight-tenths of the total time. So that with poor material it does not pay to give so much extra time for such a small additional recovery.

At Deloro (curve *E*) the gold dissolved was 14 dwt. 18 gr. per ton and the quantity obtained in the last interval was 7.78 gr. = 1s. 3½d. per ton, so that it was profitable to carry the treatment to the limit, but with a 5 dwt. material, and with conditions approaching those of curves *B* or *C*, it would be evidently unprofitable to continue the operation beyond seven-tenths of the total dissolution period.

Another set of figures illustrating this point is given by Alfred Chiddy.<sup>1</sup>

<sup>1</sup> *Jour. Soc. Chem. Industry*, vol. xix. p. 25.

As he does not give the total time required for the complete dissolving of all the accessible gold, the curve cannot be plotted in Fig. 23, but is given separately in Fig. 24, in which the same general features are apparent, namely, a rapid rise at first, followed by a sharp curve, which is succeeded by a slightly inclined straight line.

In Fig. 25 a curve has been plotted showing the rate of dissolution of

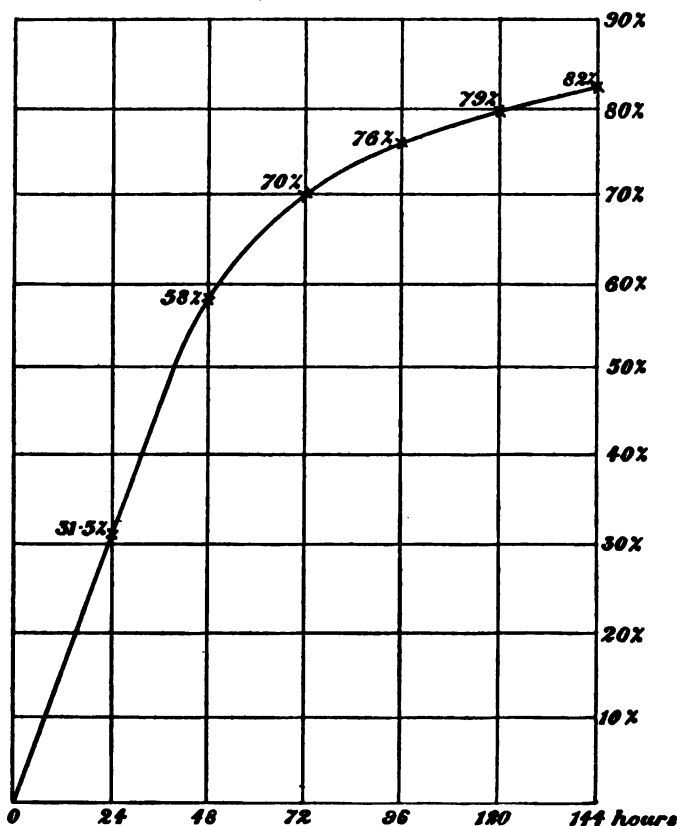


FIG. 24.—Rate of Solubility of Gold in an Ore.

*Note.*—In this diagram the percentages are calculated on the assay of the ore.

silver in a cyanide solution which was made up to strength every four hours. The test was made by Francisco Narvaez,<sup>1</sup> who gives percentages of original assay content.

The percentages have been recalculated on the basis of total silver dissolved, so that this curve is comparable with the others. It is of the same general shape as the Deloro curve in Fig. 23, although slightly flatter.

Narvaez says that, theoretically, the amount dissolved will be proportional to the surface of silver mineral exposed to the solution, as assumed at the

<sup>1</sup> *Eng. and Min. Jour.*, Nov. 21, 1908.

beginning of the chapter, and as this will be proportional to the squares of the diameters of the particles as they dissolve, the curve must be a parabola.

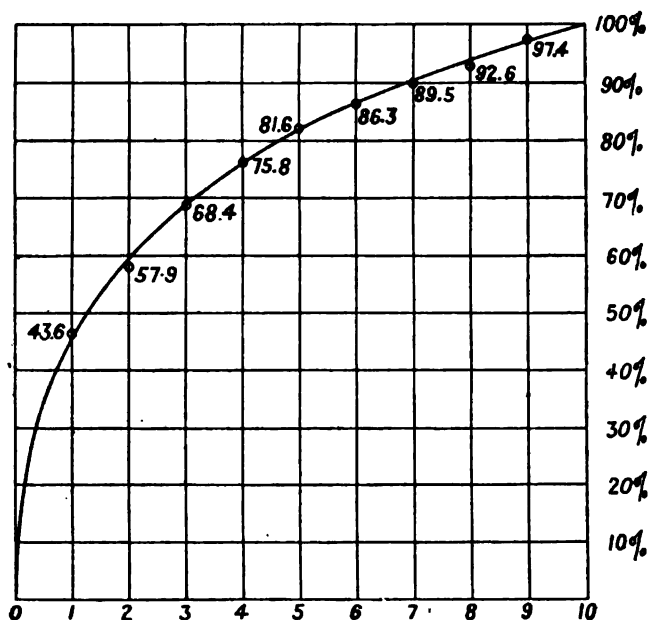


FIG. 25.—Curve plotted from Narvaez's figures for the dissolving of Silver.

This last assumption is, however, only strictly true when the particles are all of the same size and of spherical or other isometric form, such as the tetrahedron, cube, or octahedron.

## CHAPTER XI.

### OXIDISING AND DEOXIDISING OF CYANIDE SOLUTIONS.

**Solubility of Gases in Liquids.**—Gases that do not act chemically on each other or on a solution, dissolve in proportion to the pressure of the gas, and when the pressure is lowered or the temperature raised they are expelled. From this it is thought that the process of solution is purely a mechanical one. In general, however, gases dissolved in salt solutions do act chemically to a certain extent. The solvent exerts a selective influence and dissolves more of one gas than another; and when two gases are mixed, each dissolves in proportion to the partial pressure it exerts, combined with its own specific solubility in the solution. Thus air dissolved in water at freezing point and at normal atmospheric pressure contains about 35 per cent. oxygen, whereas air in the atmosphere contains about 21 per cent.

**Solubility of Air in Water.**—When air passes into water, oxygen and nitrogen dissolve until a point is reached at which the number of molecules entering the solution from without in a given time is equal to the number of molecules that disperse. At this point the solution is said to be saturated. The quantity of each gas dissolved is dependent solely on the pressure exerted by the particular gas, and the specific solubility is unaltered by presence of other gases. Thus Bunsen found that the absorption coefficient of oxygen at 0° C. equals 0.04114, and that of nitrogen equals 0.02035. Now, air contains 79 per cent. nitrogen to 21 of oxygen by volume. Hence the partial pressure of the oxygen equals 0.21 of an atmosphere, and that of nitrogen equals 0.79.

Therefore  $0.21 \times 0.04114 = 0.0086394$ , the proportion of oxygen dissolved, and  $0.79 \times 0.02035 = 0.0160765$ , the proportion of nitrogen dissolved.

According to these figures, the percentage composition by volume of air dissolved in water at 0° C. is, in round numbers, 65 nitrogen to 35 oxygen.

It is obvious that if the absorption coefficients at other temperatures are known, the proportion of each gas dissolved from air can be found.

Table XV. gives Bunsen's<sup>1</sup> absorption coefficients of oxygen in water from an atmosphere of the gas at different temperatures when the barometer indicates 760 mm. pressure. The figures represent the volume of gas dissolved by one volume of water after the gas had been reduced to 0° C. and 760 mm. pressure. The oxygen coefficients have been calculated from air, and placed in the adjoining column.

The ratio of O coefficient from O to the O coefficient from air is approximately 5 : 1.

<sup>1</sup> Bunsen's *Gasometry*.

TABLE XV.

| Temperature. |      | Bunsen's O<br>Coeff. from O. | O Coefficient<br>from Air. |
|--------------|------|------------------------------|----------------------------|
| C.°          | F.°  |                              |                            |
| 0            | 32   | 0·04114                      | 0·008639                   |
| 2            | 35·6 | 3907                         | 8205                       |
| 4            | 39·2 | 3717                         | 7805                       |
| 6            | 42·8 | 3544                         | 7442                       |
| 8            | 46·4 | 3389                         | 7116                       |
| 10           | 50   | 3250                         | 6825                       |
| 12           | 53·6 | 3133                         | 6574                       |
| 14           | 57·2 | 3034                         | 6371                       |
| 16           | 60·8 | 2949                         | 6192                       |
| 18           | 64   | 2884                         | 6056                       |
| 20           | 68   | 2838                         | 5959                       |

From Winkler's<sup>1</sup> results the weights of oxygen dissolved in water saturated with air have been calculated in grains per ton of 2000 lb. at 760 mm. pressure and at various temperatures. These are given in Table XVI.

**Solubility of Air in Salt Solutions.**—When a salt is dissolved in water the coefficient of absorption of air is generally smaller, but in cyanide working solutions only small percentages of salts are present, which do not materially affect the amount capable of being absorbed. Thus Maclaurin<sup>2</sup> found that the oxygen coefficient of absorption by a 1 per cent. KCN solution

TABLE XVI.

| Temperature. |      | Grains of O<br>dissolved in<br>1 ton Water. | Temperature. |      | Grains of O<br>dissolved in<br>1 ton Water. |
|--------------|------|---|--------------|------|---|
| C.°          | F.°  |   | C.°          | F.°  |   |
| 0            | 32   | 20·37                                       | 16           | 60·8 | 13·78                                       |
| 2            | 35·6 | 19·28                                       | 18           | 64·4 | 13·23                                       |
| 4            | 39·2 | 18·28                                       | 20           | 68   | 12·7  |
| 6            | 42·8 | 17·36                                       | 22           | 71·6 | 12·22                                       |
| 8            | 46·4 | 16·52                                       | 24           | 75·2 | 11·77                                       |
| 10           | 50   | 15·74                                       | 26           | 78·8 | 11·34                                       |
| 12           | 53·6 | 15·03                                       | 28           | 82·4 | 10·92                                       |
| 14           | 57·2 | 14·38                                       | 30           | 86   | 10·51                                       |

was 0·0280 at 18° C., whereas that absorbed by water at the same temperature, according to Bunsen, is 0·02884. When, however, sea-water is employed, or, as in W. Australia, mine water containing 30 per cent. common salt, the absorption coefficient may be materially affected. Thus, from Maclaurin's figures a 5 per cent. KCN solution reduces the coefficient to 0·0230, and a 30 per cent. KCN to 0·0079. A reduction in the coefficient means a reduction in

<sup>1</sup> *Berichte d. d. chem. Gesell.*, xxii. 1773.

<sup>2</sup> *Jour. Chem. Soc.*, vols. lxvii. and lxviii. p. 211.

the amount of available oxygen for purposes of oxidising reducing agents in the solution and ore and of oxidising hydrogen as formed in the process of dissolving the metals.

When the solution is made thick and viscous by means of slime or sand, a marked effect is noticeable on the coefficient of absorption, although the amount of dissolved salt is very minute. Thus, when the pulp consists of two parts of solution (0.05 per cent. KCN) to one part of slime, the coefficient of absorption of air is less than one-hundredth of that of the solution alone.

Another point of practical importance is the time required to saturate the solution with air. If air is removed from a solution by heating or otherwise, the time required to recharge the solution varies with the viscosity of the solution. An ordinary cyanide solution takes longer than water, and slime pulp takes longer than simple cyanide solution. It often happens that the oxygen absorbed from the air becomes removed by reducing agents, leaving the nitrogen intact; so that, in order to replace the oxygen, air has to be passed into the solution for a considerable time. As a substitute, and to shorten the time of treatment, strong oxidisers are sometimes beneficially added to the pulp.

This increase in the time required to saturate a solution with air points to an increased friction to the gas molecules which retard their diffusion. A rise in temperature lessens the viscosity, and consequently allows a faster rate of diffusion, but rise in temperature diminishes the coefficient of absorption. It is evident, then, that the time of saturation is dependent on these two factors.

**Deoxidising of Cyanide Solutions.**—The subject of the deoxidising of cyanide solutions is dealt with in the chapter on "Precipitation by Zinc."

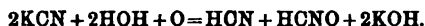
## CHAPTER XII.

### SOURCES OF LOSS OF CYANIDE.

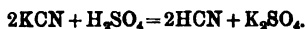
**Decomposition of the Solution.**—An alkaline cyanide is very easily decomposed by any mineral acid, and less easily by salts of these acids. Even water decomposes single cyanides by hydrolytic action. Potassium cyanide is an inodorous substance in an atmosphere free from moisture, but when moisture is present the well-known odour of HCN is at once recognised. The moisture acting as an acid produces the following chemical changes—



and in presence of dissolved oxygen



Hydrocyanic acid, being only slightly soluble in water, and generally less soluble in solutions of salts, soon begins to escape into the atmosphere. True, the decomposition of KCN in this way is infinitesimally small as compared with the action of other acids. Even a weak acid, such as carbonic acid, has a stronger action by thousands of times, while the highly dissociable acids, as HCl,  $\text{NHO}_3$ , and  $\text{H}_2\text{SO}_4$ , have such powerful actions that in a moderately concentrated state they will split up even such a slightly dissociable substance as cyanogen, forming compounds not belonging to the cyanogen group. If, however, the acid is dilute, such as is found present in partly decomposed ores, the action is similar to that which occurs with water, thus—



In very dilute solutions the HCN escapes into the air so very slowly that, even some hours after the solution became acid, practically the whole of the HCN is still present, and if alkali is added, it will be found that the solution has lost little of its original strength. Some HCNO or KCNO is also formed in presence of dissolved air.

It often happens on heating KCN solutions much above  $140^\circ \text{F.}$ , and even at lower temperatures, formates and acetates are formed, and a number of complicated changes occur which involve loss of cyanide. The reactions are, however, very uncertain, and it sometimes happens that the solution may be raised even to boiling point without showing any appreciable chemical change.

In the absence of reducing agents, oxygen absorbed from the air slowly oxidises the cyanide to cyanate of cyanic acid, as shown above. In working solutions, traces of reducing agents are nearly always present, so that the loss



from this cause is very small. Oxidising agents, such as  $\text{KMnO}_4$ ,  $\text{K}_2\text{Fe}(\text{CN})_6$ ,  $\text{PbO}_2$ , readily oxidise cyanide to cyanate, but in working solutions, and especially in presence of reducing agents, this action is not marked.

Vasculose readily destroys cyanide in absence of an excess of alkali. Coconut fibre, jute, and wood are especially destructive when new, but after having been in contact with the solution some time they lose this effect. Vasculose acts as a reducing agent in a cyanide solution.

Carbonic acid is absorbed by cyanide solutions from the atmosphere, and in absence of excess of free alkali  $\text{HCN}$  is evolved. In moderately still air a square yard of surface absorbs 1000 c.c. of  $\text{CO}_2$  per hour, and in a strong wind or by constantly stirring it absorbs 1200 to 1300 c.c. per hour. The rate of absorption is hardly affected by the amount of alkali present within working limits, but the tension of the  $\text{CO}_2$  has a very marked influence. The figures given refer to cases where the top of the tank is several feet above ground level. But when the top of the tank is on the level or below the level of the surrounding ground, the rate of absorption may be three or four times as fast.

The presence of soluble and insoluble carbonates facilitates decomposition of cyanides.

**The Action of Cyanide on the Constituents of the Ore.**—The cyanide process is only made possible by the fact that the action of the solution on the ore constituents, compared with that on gold and silver, is for the most part but slight. That is to say, that the gold and silver are electro-positive in a cyanide solution to the other matter with which they are associated. Many constituents of ores are, however, appreciably soluble in cyanide solutions, and as they expose generally very large surfaces compared to that of the gold and silver, it often happens that larger quantities are dissolved.

**Metallic Iron or Steel** is usually introduced into the ore pulp in the process of crushing and becomes disseminated throughout the mass in fine grains. It is also employed in the construction of parts of the plant. Iron in a  $\text{KCN}$  solution dissolves slowly and is much less soluble than gold or silver, as shown in Fig. 19. The final reaction is probably expressed by the equation—



The hydrogen becomes oxidised to water, as formed, by oxygen or oxidising agents in the solution.

The actual weight of iron introduced into the ore or brought into contact with the solution is far in excess of the gold and silver usually present, but the gold and silver being in a finer state of division and being more soluble, it is probable that a larger weight of these metals dissolves during the time of treatment. But it must be remembered that 56 parts of iron consume 390 parts of  $\text{KCN}$ , whereas 197 of gold consume only 130 parts of  $\text{KCN}$ . That is to say, 1 lb. of iron requires 7 lb. of  $\text{KCN}$  for dissolution, whereas 1 lb. of gold requires only 0.66 lb.  $\text{KCN}$ , or less than one-tenth.

Iron not only consumes much cyanide, but it also consumes oxygen required for the dissolution of the gold and silver, and it follows from the

reactions that 56 parts of iron will consume twice as much oxygen as 197 parts of gold.

Ferric oxide in electrical contact with gold in a cyanide solution acts as a depolariser, and produces a lower oxide that is more soluble in the solution.

**Iron Sulphides** are generally present in gold ores in two well-known forms, as marcasite (the orthorhombic mineral) and pyrite (the isometric mineral); both have the same composition, and are represented by the formula  $\text{FeS}_2$ . Pyrites is the general name for them.

Marcasite is of a paler and duller colour and lower specific gravity (4.8) than pyrite; also it is well known to have a greater tendency to oxidise in moist air, with formation of  $\text{FeSO}_4$ . Pyrite is of a brassy yellow colour, resists atmospheric oxidation much more than marcasite, and has a specific gravity of about 5. It is very common to find both minerals associated together in gold ores.

In a cyanide solution, marcasite is much less soluble than pyrite, as may be seen on reference to Fig. 19. The reaction appears to be doubtful, but the chief compounds formed are  $\text{K}_4\text{Fe}(\text{CN})_6$ ,  $\text{KCNS}$ ,  $\text{K}_2\text{S}$ , and  $\text{K}_2\text{S}_2\text{O}_8$ . From several experiments made, it is evident a considerable quantity of oxygen is consumed in the reaction, as it is found that pyrites is hundreds of times more soluble when a large excess of oxygen is present. Unoxidised pyrite appears to be insoluble in absence of oxygen, at atmospheric pressure and ordinary temperatures. With an excess of oxygen the pyrite causes the solution to become coloured, varying in shade from an amber tint to a muddy brown. It loses colour by standing when exposed to the atmosphere and sunlight for some days, or by the use of oxidising and desulphurising agents. This brown solution was found to have lost much of its dissolving power on gold and silver, but the power is easily restored by the use of desulphurising and oxidising agents.

It has long been observed that when marcasite has been allowed to weather, it heats and rapidly oxidises, forming  $\text{FeSO}_4$  and  $\text{H}_2\text{SO}_4$ . On the other hand, pyrite oxidises slowly to limonite,  $\text{Fe}_2\text{O}_3(\text{OH})_2$ , and sulphur for the most part, with but little  $\text{FeSO}_4$ . Caldecott<sup>1</sup> says that Lœvy proved that when  $\text{FeS}_2$  is gradually oxidised, the first compound formed is  $\text{SO}_2$ , also that  $\text{H}_2\text{S}$  is formed through the action of  $\text{H}_2\text{SO}_4$  on  $\text{FeS}$ , the latter being formed by the oxidation of one atom of sulphur of the  $\text{FeS}_2$ .

The presence of  $\text{FeSO}_4$  in an ore is the cause of a considerable consumption of cyanide. It is soluble in water, and if the ore is leached with clean water a portion may be removed, which thus saves cyanide. Some, however, always remains behind, and this becomes neutralised on the addition of alkali, forming ferrous hydrate and alkaline sulphate. Ferrous sulphate and hydrate act as deoxidisers, and consume oxygen that might otherwise be of service in dissolving the gold and silver. Ferrous sulphate oxidises to normal and basic ferric sulphates. The former is soluble in water, and the

<sup>1</sup> *S. African Min. Jour.*, Oct. 23, 1897, from paper read before S. Af. Chem. and Met. Soc.

latter is soluble in a solution of the former. They are both acted on readily by KCN solutions, forming, amongst other compounds,  $K_4Fe(CN)_6$ , some Prussian blue, and probably  $K_3Fe(CN)_6$ . Metallic iron in the ore reduces some ferric sulphate to ferrous salts.

Ferrous hydrate is readily oxidised to ferric hydrate, a name applied to many indefinite compounds, having the formula  $Fe_2O_3 \cdot xH_2O$ , of which the exact composition varies. Some of these compounds are insoluble in KCN, but others dissolve more or less readily, probably forming  $K_4Fe(CN)_6$ , and acting to some extent as oxidisers.

Limonite, a mineral often associated with gold ores, is a hydrated oxide of iron. It does not appear to dissolve in KCN to any appreciable extent, but often forms unstable suspensions that become troublesome in the process of leaching.

Copper in the metallic state is found occasionally associated with gold ores, and usually it is introduced during the mining operations in the form of detonator cases. The final reaction when copper is dissolved in KCN is generally stated as



Therefore 1 part of copper requires 2 parts KCN for solution, or three times as much as gold. The solution also loses oxygen by its combining with some of the hydrogen formed at the negative electrode.

Copper sulphides are invariably present in gold-bearing ores, the amounts varying from a trace up to the proportions of a rich copper ore. Copper pyrite is perhaps the most commonly met with. Erubescite, covellite, and redruthite are less often present.

These minerals are all acted on by cyanide solutions, but some are much more soluble than others. Tests to determine the solubility of a mineral by itself are of doubtful value, as much depends on the other minerals with which it is in contact. Thus copper pyrite, which by itself dissolves slowly in KCN, when in contact with a less positive mineral, as marcasite, dissolves much more rapidly. Then again, when iron pyrite is present and becomes oxidised to ferric sulphate, most copper sulphides are also oxidised to sulphate. The copper sulphate remains in solution, but when this compound comes into contact with metallic iron, the copper precipitates, and this precipitate readily dissolves in cyanide solutions.

The presence of copper is not necessarily a bar to the treatment of an ore by cyanide, for it sometimes happens that when even 2 or 3 per cent. of copper is present an ore can be treated profitably by cyanide, but not by other processes. On the other hand, sometimes an ore containing less than 0.5 per cent. copper could not be treated profitably, owing to the destruction of cyanide. A mere chemical analysis of an ore is not to be relied on as an absolute guide in testing copper-bearing ores, as much depends on the physical conditions in which the copper and gold exist.

There are a large number of minerals containing copper in some form associated with ores that carry both gold and silver, and it sometimes happens

that the gold and copper are dissolved, but scarcely any of the silver, whereas in others the gold is practically the only metal dissolved. It is probable that the copper and silver are sometimes in chemical combination, as in the minerals stromeyerite,  $\text{Cu}_2\text{S}, \text{Ag}_2\text{S}$ , and jalpaite,  $\text{Cu}_2\text{S}_3, \text{Ag}_2\text{S}$ , and at others are mixtures of two or more simple compounds, as  $\text{CuS}$  and  $\text{AgS}$ . The solubility of silver is usually small when copper is present, whereas the solubility of gold is seldom affected to the same extent.

Occasionally the minerals malachite,  $\text{CuCO}_3, \text{CuH}_2\text{O}_2$ , and azurite,  $2\text{CuCO}_3, \text{CuH}_2\text{O}_2$ , are found associated with gold and silver ores. These consume so much cyanide that ores containing even very small quantities of either become unprofitable for cyanide treatment.

**Native Arsenic** is occasionally found associated with gold ores, or is reduced from its salts in the process of crushing. Arsenic does not interact with potassium cyanide nor with alkali in the solution, but nearly all compounds of arsenic found in gold ores are soluble in caustic alkali, and decompose KCN in absence of free alkali.

**Mispickel** (arsenical pyrites),  $\text{FeAs}, \text{FeS}_2$ , is, next to iron pyrite, perhaps the most abundant metallic compound found associated with gold ores, and is in general the more auriferous of the two. Mispickel is acted on by cyanide, its relative solubility to gold being small, as shown in Fig. 19. It weathers in moist atmospheres, for the most part to iron sulphate and to hydrate and oxide of arsenic. The latter does not combine with the cyanide, but in absence of free alkali decomposes the solution, liberating HCN and combining with the alkali.

In treating mispickel ores, it is generally found that the addition of large amounts of lime to the ore effects a considerable saving in cyanide and improves the extraction. If magnesia is procurable it will be found to answer equally well, and as a rule a higher extraction can be relied on.

Sulphides of arsenic, as realgar and orpiment, are both attacked by alkalis, forming arsenites and thio-arsenites. Alloys of iron and arsenic, as arsenical iron, are occasionally rich in gold and silver. These compounds are little acted on by cyanides, but the gold and silver are not readily attacked unless the mineral is reduced to a fine state of division.

The compounds of alkali and arsenic that form act as reducers, and consume oxygen dissolved in the cyanide solution, and in this way retard the dissolution of the gold and silver.

**Antimony Sulphide**, as stibnite, is often found associated with gold ores, and is sometimes highly auriferous. Like arsenic, antimony does not form any definite compound with cyanide, but the sulphide is very soluble in caustic alkali and decomposes cyanide, combining with the alkali, and forming antimonite and thio-antimonite; also KCNS is formed and HCN is evolved. The antimony compounds act as strong deoxidisers, and remove the absorbed oxygen from the solution by forming antimonate and thio-antimonate. Gold and silver may be rendered almost insoluble from this cause. Much of the cyanide is destroyed.

Other antimony minerals, as antimony bloom, senarmontite, and antimony ochre, occur in gold ores, but much less frequently than stibnite, and are less difficult to treat.

**Tellurium** is often found in minerals associated with gold ores, which are usually highly auriferous and argentiferous. In some minerals it would appear to be in the form of an alloy with the gold or silver, or in more or less definite compounds, as calaverite, krennerite, petzite, and sylvanite, forming tellurides of gold and silver. Nagyagite, or foliated tellurium, is a complex substance, containing, besides gold and silver, varying quantities of sulphur, lead, copper, and antimony. Gold is sometimes found in such minerals as hessite and melonite, and tellurium is also found in auriferous arsenical iron, mispickel, iron and copper pyrites.

The presence of tellurides, when forming any appreciable proportion of the metallic minerals, usually makes an ore difficult to treat, as the action of the cyanide on the gold is slow. The cause of this slowness of action is not altogether apparent, but it is evident that there is a wide difference in the solubility of the gold in different telluride minerals, and even in the same minerals from different localities. Thus, in Western Australia, where the gold is in a sulpho-telluride, the ore is capable of being dealt with by fine grinding and long treatment, whereas the same treatment applied to certain Mexican and United States tellurides is less successful.

There is no action between the cyanogen radical and the tellurium, but the alkali in presence of oxygen invariably dissolves some of the metal, and also acts on the sulphur when in combination, forming a solution which has a reducing action. When a telluride ore is roasted, it leaves a residue containing  $\text{TeO}_2$ , and this oxide is very soluble in  $\text{KOH}$ , forming a tellurite, which also acts as a reducing agent and absorbs oxygen from the solution. The same change takes place with  $\text{KCN}$ , with evolution of  $\text{HCN}$ . Roasted tellurides are, however, capable of being treated and the gold extracted with good results.

**Mercury**, as cinnabar,  $\text{HgS}$ , is sometimes found associated with gold and silver ores. Its action on cyanide solution is *nil*. Mercury has been found in Kalgoorlie ores as coloradoite, which is readily attacked by cyanide. Metallic mercury may be present in tailings, being introduced from the amalgamation process, and is sometimes in considerable quantity. In a well-managed mill the loss in mercury usually varies from 0.01 to 0.03 lb. per ton of ore crushed, but this is often exceeded when much base mineral is present. Theoretically, the whole of this finds its way into the cyaniding vats, and is acted on by the cyanide solution.

Mercury is somewhat less soluble than gold in cyanide, and probably combines according to the following equation—



Oxygen is required for the dissolution, in order to combine with the hydrogen as formed, and this is taken from the oxygen absorbed by the solution. Thus

200 parts of mercury requires 260 parts of KCN for solution, that is, twice as much as gold. Twice as much oxygen is also needed.

Mercury has, however, a far greater affinity for sulphur than for cyanogen, and as the solutions usually contain sulphides, thiosulphates, and thiocyanates, these or the sulphur in them combine with mercury to form  $\text{HgS}$  and a number of complex compounds, amongst which are  $\text{HgSCN}$ ,  $2\text{KCNS}$  and  $\text{Hg}(\text{CN})_2$ ,  $\text{KCNS}$ , which do not appear to retard the dissolution of the gold and silver. Mercury, therefore, acts as a desulphurising agent, and although it consumes cyanide, its presence is not altogether undesirable.

Zinc, as zinc-blende, is not commonly met with in gold ores. When taken fresh from the mine in an unoxidised condition it is only slightly acted on by KCN, forming some  $\text{KCNS}$  and probably some complex substances. Blende from some mines is much more soluble than from others, and this appears to be somewhat dependent on the minerals associated with it. If partly oxidised, such as happens when ore weathers, blende destroys both cyanide and alkali.

Galena is often associated with both gold and silver ores. In a clean unoxidised condition it is only slightly acted on by KCN, but with long contact  $\text{KCNS}$  is formed. When partly oxidised by atmospheric influences, long contact with KCN produces evolution of  $\text{HCN}$ .

Minerals taken from different localities may vary in the way they are acted on by KCN solutions, as a great deal is dependent on the other minerals with which they are associated. Thus, marcasite or pyrite acted on separately have comparatively small effect on a KCN solution, but when the two minerals are mixed and dealt with together the destruction of cyanide is much increased.

Again, many of the complex compounds formed with the KCN are decomposed in the precipitating boxes, the metal being deposited or forming insoluble compounds which precipitate, while others are rendered less soluble, and probably become to some extent precipitated in the ore.

## CHAPTER XIII.

### ACTION OF VARIOUS CYANIDE SOLUTIONS.

**Relative Dissolving Power of Cyanides.**—The weights of simple cyanides dissolved in equal quantities of water, so as to have the same dissolving effect on gold or silver, are dependent on the ratio of the valency of the base and the molecular weight of the salt. Thus sodium and potassium have a valency of 1. They combine with cyanogen to form NaCN and KCN, which have molecular weights of 49 and 65 respectively. Therefore it will require only 49 parts by weight of NaCN to produce the same dissolving effect as 65 parts of KCN. But in the case of calcium the valency is 2 and the molecular weight of  $\text{Ca}(\text{CN})_2$  is 92. Hence it will require  $92 \div 2$  or 46 parts to have the same dissolving effect as 49 of NaCN or 65 of KCN.

In Table XVII., Column 1 gives the formula of various simple cyanides; 2, their molecular weights; 3, valency of the base; 4, relative weights dissolved in equal quantities of water to give the same dissolving effect; and 5, relative dissolving power of the same weights of the salts in terms of KCN at 100.

TABLE XVII.

| 1                        | 2     | 3 | 4     | 5     |
|--------------------------|-------|---|-------|-------|
| $\text{NH}_4\text{CN}^1$ | 44    | 1 | 44    | 147.7 |
| NaCN                     | 49    | 1 | 49    | 132.6 |
| KCN                      | 65    | 1 | 65    | 100   |
| $\text{Mg}(\text{CN})_2$ | 76    | 2 | 38    | 171   |
| $\text{Ca}(\text{CN})_2$ | 92    | 2 | 46    | 141.3 |
| $\text{Sr}(\text{CN})_2$ | 139.5 | 2 | 69.75 | 93.2  |
| $\text{Ba}(\text{CN})_2$ | 189   | 2 | 94.5  | 69.3  |

The stability of these compounds varies greatly, KCN being the most stable, and  $\text{Ca}(\text{CN})_2$  the least. The following is the order of stability in an atmosphere containing  $\text{CO}_2$ —

KCN, NaCN,  $\text{NH}_4\text{CN}$ ,  $\text{Mg}(\text{CN})_2$ ,  $\text{Ba}(\text{CN})_2$ ,  $\text{Sr}(\text{CN})_2$ ,  $\text{Ca}(\text{CN})_2$ .

**Dissolving Power of Potassium Zinc Cyanide.**—The comparative dissolving power of potassium zinc cyanide and potassium cyanide has been much discussed. It was thought by many in the early days of the cyanide process that the  $\text{K}_2\text{Zn}(\text{CN})_4$ , formed in the precipitating of the gold by zinc, was use-

<sup>1</sup>  $\text{NH}_4\text{CN}$  has an abnormal dissolving effect on silver, that given being for gold.

less for dissolving any further quantity of gold from the ore. It was, however, soon pointed out that  $K_2Zn(CN)_4$  had a solvent action on gold in ores.

G. A. Goyder<sup>1</sup> prepared some pure  $K_2Zn(CN)_4$ , and found that gold dissolved in it in presence of oxygen with the production of gold potassium cyanide and oxide of zinc. A number of other authorities have also shown that  $K_2Zn(CN)_4$  dissolves gold.

Julian made experiments to compare the dissolving effects of KCN and  $K_2Zn(CN)_4$  on gold. Some 98 per cent. KCN was dissolved in water, and its dissolving effect on gold was determined. ZnO was added to a portion of the same solution in excess until it was probably saturated with zinc, and the dissolving effect of this solution on gold was also determined. These were found to be in the ratio of 109 : 96 respectively. A large excess of alkali was next added to both solutions, and their dissolving effects were found with KCN to have increased to 133, and with  $K_2Zn(CN)_4$  to 128. It is clear from these results that the presence of zinc in the solution is not at all times detrimental.

**Cyanide Solutions in Practice.**—It is usual in the cyanide process to regard the solution as consisting of potassium cyanide, whereas in reality it is a mixture of a number of substances which form a solution of a very complex nature. Some of these substances play no part in the dissolution of the gold and silver, whereas others accelerate or retard the process, either directly or indirectly.

The presence of practically inert substances, such as  $K_2SO_4$ , or KCNS, can cause little or no increase in the solubility of the gold and silver, but has rather the reverse action, through increasing the viscosity of the solution, and thereby decreasing the velocities of the K and CN ions of the KCN salt.

In the great majority of cases no undue accumulation of zinc occurs in a cyanide solution as a result of the use of this metal as the precipitant. In other instances the zinc content rises above the economic limit, and a high zinc and cyanide consumption results. In cases where it is practicable to keep cyanide solutions with a high free cyanide content, an excess of zinc in the solution is unnoticeable and seldom affects its dissolving power. It does, however, affect the efficiency of precipitation and invariably leads to an excessive consumption of both solvent and precipitant. In cases where solution or precipitation of the gold or silver is injuriously affected by the presence of zinc it is advisable to reduce the zinc content, or to substitute an alternative compound which under working conditions forms no undesirable compounds with cyanide.

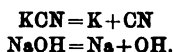
In some cases reducing agents have been known to accumulate to such a degree as to prove very detrimental, by retarding the dissolution of both gold and silver. When the KCN solution is weak, the presence of small quantities of soluble sulphides has a direct effect on silver, by forming a superficial coating of silver sulphide on the particles, which renders the silver almost insoluble. With strong solutions this effect is not marked.

<sup>1</sup> *Chem. News*, vol. 72, p. 96.

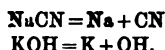


On the other hand, oxidising agents in suitable quantities have a beneficial effect and tend to keep the rate of dissolution of the metals more constant. In fact, without an oxidiser in some form, dissolution soon ceases under ordinary atmospheric changes. Then, again, the presence of substances in which  $\text{AgCN}$  and  $\text{AuCN}$  are soluble, such as thiosulphates of the alkalis or ammonia, tends to increase the solubility of gold and silver in cyanide solutions, but these substances sometimes interact with the cyanide, so that their value may be considered as doubtful.

The solubility of gold in alkaline cyanides varies with the base. The amount of gold that will dissolve in a given time in a 1 per cent. KCN solution will be different from the amount that will dissolve in a 1 per cent. NaCN solution, but if normal solutions or equally diluted normal solutions are considered the effects will be the same. When equal quantities of normal solutions of KCN and NaOH are mixed and moderately diluted, they will have exactly the same action as normal solutions of NaCN and KOH diluted to the same extent. In dilute solutions these compounds become completely dissociated into ions thus—



and in the second case,



It will be seen that, in both cases, the same K and Na cations and the same CN and OH anions are formed. It has been proved that the action of dissociated solutions depends only on the ions, and as the ions are alike, the action in both cases must be the same.

When two solutions, which do not alter the properties of each other, such as KCN and NaCN, are mixed, a metal dissolves in them, and divides itself between the two solutions in the ratio of their respective ions. Thus, in dissolving gold in a mixture of KCN and NaCN,  $\text{KAu}(\text{CN})_2$  and  $\text{NaAu}(\text{CN})_2$  are formed, in proportion to the ions present of the respective salts.

**Stability Test.**—When making a selection from various brands of cyanide, the stability should always be compared. This may be conveniently done by making a solution (3 litres 0.5 to 1 per cent.) of each brand. Place in beakers of the same size and allow them to stand uncovered, where there is a free circulation of air, for about a week. Test again, and note the loss. Some brands of cyanide lose 20 per cent. while others lose only 0.5 per cent. during this time. A loss of not more than 1 per cent. is considered satisfactory. It should be remembered that traces of reducing agents in the solution increase the stability, but decrease the dissolving effect. This test is therefore not conclusive in making the selection.

**Dissolving Effect Test.**—Prepare similar samples, as in the stability test; suspend, horizontally with threads, in each a piece of clean platinum or iron-wire gauze, 3 in. in diameter, a few inches below the surface of the solution. On this spread, say, four  $\frac{1}{2}$ -in. squares of fine-gold foil (annealed), previously

weighed; allow to stand for a week in a cool place of even temperature, then remove the gold squares, wash, dry, and re-weigh. The difference in weight will indicate the difference in dissolving effect of the solutions.

The solution that shows the least percentage loss in cyanide per unit weight of gold dissolved, gives a fair indication of the best brand to select.

Thus in the stability test if

$$\begin{array}{lll} x = \text{per cent. of KCN in the solution at beginning,} \\ x' = & \text{,,} & \text{,, end of week,} \end{array}$$

and in the dissolving effect test if

$$\begin{array}{lll} y = \text{weight of metal at beginning,} \\ y' = & \text{,,} & \text{,, end of week,} \end{array}$$

then the solution of which  $\frac{x-x'}{y-y'}$  is least, indicates the brand to select.

## CHAPTER XIV.

### THEORY OF THE PRECIPITATION OF GOLD AND SILVER.

#### *Section I.*

**Electrolytic Methods.**—It is customary to consider precipitation by an electric current, applied from an external source, as being essentially different from precipitation by zinc or other metal. On investigation, however, it will be found that the factors governing precipitation are similar in each case.

When two metals are immersed in a liquid having a greater chemical action on one than on the other, a difference of electric potential, or e.m.f., is set up; and when the two metals are connected either within or without the liquid, an electric current is produced. Even a single metal, whose physical condition varies at different parts of its surface, is capable of producing an electric current through the liquid from one part to another. Any combination of substances producing this effect is known as a galvanic couple, the more soluble of these substances being termed the anode and the less soluble the cathode.

In precipitation, a certain minimum e.m.f. is required, in amount varying with different metals; with the same metal in different solutions; and to some extent with the strength of the solution. When the current is supplied from an external source, such as by dynamo or battery, the e.m.f. may be regulated to a nicety, and measured with great accuracy. On the other hand, when the action of a galvanic couple is taken advantage of in a precipitation process the same facility does not exist for current regulation; and with some metals the e.m.f. produced is not great enough to effect precipitation. Silver containing lead as an impurity will generate a current in a cyanide solution, but the e.m.f. produced is not high enough to precipitate gold from such a solution. Each metal has its own potential, relative to the liquid, being either positive or negative; and the effective difference of potential for two metals, that is to say, the e.m.f. of the combination, is the algebraic difference of the two potentials. If the two potentials have the same sign, the smaller potential must be deducted from the greater, but if of the opposite sign, they must be added together to give the e.m.f. The minimum e.m.f. required to precipitate gold or silver varies with the strength of the solution, as already pointed out. It is also influenced by the temperature and the physical condition of the cathode.

When the gold is precipitated it asserts its own potential, and tends to redissolve, producing an e.m.f. which drives a current in the opposite direc-

tion to that of the precipitating current. The result is that the e.m.f. of the latter is reduced by that of the gold. Thus Christy found zinc to have a potential of  $+0.82$  volt, and lead to have a potential difference of  $+0.05$  volt, with respect to a 0.65 per cent. KCN solution. Therefore the e.m.f. of a zinc-lead couple in that solution equals  $0.82$  minus  $0.05$ , that is,  $+0.77$  volt. The lead in time becomes covered with gold; and as the potential difference of gold, with respect to the same solution, was found to be  $+0.23$ , the e.m.f. of the current falls to  $0.82$  minus  $0.23$ , or  $+0.59$  volt. The full benefit of the  $0.77$  voltage is not obtained, for at the moment gold deposits on the lead a gold-lead couple is formed, whereby the gold tends to redissolve with a varying e.m.f., which ultimately becomes  $0.23$  minus  $0.05$ , or  $0.18$  volt. This is small, comparatively, and for that reason lead is a suitable metal on which to deposit gold. On the other hand, if the negative metal was platinum, an ultimate e.m.f. of  $-0.46 + 0.23$ , or  $0.69$  volt would result, which would cause the precipitated gold to redissolve about four times as fast as in the case of lead. If the negative electrode has a higher potential than gold, as, for instance, tin, then when gold is precipitated, two couples would be acting in the same direction, i.e. zinc-gold and tin-gold; but as the difference of potential between tin and gold is  $0.24$  minus  $0.23$ , or  $0.01$  volt, it follows that a zinc-tin couple is suitable for precipitating gold from this solution. It is therefore important, when precipitating metals from solutions by galvanic couples, that the electro-negative metal should stand in the electro-chemical series as near to the metal being precipitated as practicable.

The above results refer to a 0.65 per cent. of pure potassium cyanide solution, but when a weaker or stronger solution is used, or, as in practice, a mixture of several solutions, the results vary accordingly.

**The Electro-chemical Series.**—There is a definite difference of potential between a metal, or metallic mineral, with respect to the solution in which it may be immersed. In cyanide solutions, this potential difference is not always the same, but varies with the strength, and may sometimes be positive and sometimes negative, according to the substances, as already shown by example. When a positive metal dissolves, the transition into ions is attended with a gain of energy. The converse occurs when the ions are transformed into metal (precipitated).

A. von Oettingen<sup>1</sup> and S. B. Christy<sup>2</sup> have made a large number of determinations of potential differences of substances with respect to cyanide solutions of different strengths. These are reproduced in Tables XVIII. and XIX. von Oettingen's final observations are given, but these have been roughly spaced according to the numerical values, so as to show the changes of potential more forcibly. Christy's results are reproduced and tabulated as in his original paper.

These tables are useful for making comparisons, but as the results were obtained with pure cyanide solutions, they must not be taken as strictly

<sup>1</sup> *Jour. Chem. and Met. Soc. S. Africa*, Feb. 1899, pp. 21-28.

<sup>2</sup> *Trans. Am. Inst. Min. Eng.*, Sept. 1899

true for working solutions found in actual practice. Potentials obtained as the above are necessarily, owing to experimental difficulties, the average results of a number of observations for each substance, and this accounts to some extent for differences in the figures of the two authorities.

TABLE XVIII.—*von Oettingen's Electro-motive Series of certain Metals in KCN Solutions.*

| Last Observations.             |        |                                |        |                                |        |                                |        |
|--------------------------------|--------|--------------------------------|--------|--------------------------------|--------|--------------------------------|--------|
| N. 1000 KCN = 0.0065 per cent. |        | N. 100 KCN = 0.065 per cent.   |        | N. 10 KCN = 0.65 per cent.     |        | N. 1 KCN = 6.5 per cent.       |        |
|                                |        | Fe <sub>3</sub> O <sub>3</sub> | -0.750 | Fe <sub>3</sub> O <sub>3</sub> | -0.720 | Fe <sub>3</sub> O <sub>3</sub> | -0.674 |
| Ni                             | -0.560 |                                |        |                                |        |                                |        |
| Au                             | -0.474 | Ni                             | -0.488 | Ni                             | -0.392 |                                |        |
|                                | .      |                                |        |                                |        |                                |        |
| Cu                             | -0.230 |                                |        |                                |        |                                |        |
| Ag                             | -0.200 |                                |        |                                |        |                                |        |
|                                |        | Au                             | -0.056 |                                |        |                                |        |
|                                |        | Hg                             | -0.056 |                                |        |                                |        |
|                                |        | Ag                             | -0.020 | Fe                             | -0.012 |                                |        |
| PbO <sub>2</sub>               | -0.006 |                                |        |                                |        |                                |        |
|                                |        | Fe                             | +0.022 | Hg                             | +0.024 |                                |        |
| Fe                             | +0.050 | PbO <sub>2</sub>               | +0.070 |                                |        |                                |        |
|                                |        |                                |        | Co                             | +0.118 |                                |        |
|                                |        |                                |        | PbO <sub>2</sub>               | +0.118 |                                |        |
| Pb                             | +0.120 | Pb                             | +0.120 | Pb                             | +0.128 |                                |        |
|                                |        |                                |        | Ag                             | +0.176 | PbO <sub>2</sub>               | +0.160 |
|                                |        |                                |        |                                |        | Pb                             | +0.164 |
|                                |        |                                |        |                                |        | Ni                             | +0.194 |
|                                |        |                                |        |                                |        | Co                             | +0.196 |
|                                |        | Co                             | +0.240 | Au                             | +0.218 | Hg                             | +0.200 |
|                                |        | Cu                             | +0.380 |                                |        | Au                             | +0.306 |
|                                |        | Zn                             | +0.604 | Cu                             | +0.648 | Ag                             | +0.314 |
| Zn                             | +0.480 |                                |        | Zn                             | +0.800 |                                |        |
|                                |        |                                |        |                                |        | Cu                             | +0.924 |
|                                |        |                                |        |                                |        | Zn                             | +0.940 |

Taking von Oettingen's figures for a solution of 0.65 per cent. KCN, the following are some of the combinations, with their numerical values:—

$$\begin{array}{llll}
 a \left\{ \begin{array}{l} \text{Zn} \\ \text{Cu} \end{array} \right. & \begin{array}{l} +0.800 \\ +0.648 \end{array} & b \left\{ \begin{array}{l} \text{Zn} \\ \text{Ag} \end{array} \right. & \begin{array}{l} +0.800 \\ +0.176 \end{array} \\
 & \hline
 & +0.152 & & +0.624
 \end{array}
 \quad
 \begin{array}{llll}
 c \left\{ \begin{array}{l} \text{Zn} \\ \text{Pb} \end{array} \right. & \begin{array}{l} +0.800 \\ +0.128 \end{array} & d \left\{ \begin{array}{l} \text{Zn} \\ \text{Au} \end{array} \right. & \begin{array}{l} +0.800 \\ +0.218 \end{array} \\
 & \hline
 & +0.672 & & +0.582
 \end{array}$$

From the theoretical point of view the zinc-lead couple is capable of developing the highest e.m.f. in the series, and the zinc-copper couple, the lowest; but these comparisons only apply to the permanent conditions obtaining after a certain amount of gold or silver has been precipitated. As a matter of fact

the opposing e.m.f. of the precipitating metal is indefinitely small at the first moment of its precipitation from the solution; and in intensity it does not approach the amount indicated in the tables until the gold is sufficiently thick to take a massive form.

TABLE XIX.—*Christy's Electro-motive Series of Metals and Minerals in KCN Solutions.*

|                                  | $\frac{N}{1} \text{ KCN}$<br>= 6.5 p. ct.<br>volts. | $\frac{N}{10} \text{ KCN}$<br>= 0.65 p. ct.<br>volt. | $\frac{N}{100} \text{ KCN}$<br>= 0.065 p. ct.<br>volt. | $\frac{N}{1000} \text{ KCN}$<br>= 0.0065 p. ct.<br>volt. |
|----------------------------------|---|--|--|--|
| Aluminium, . . . . .             | +0.99   | +0.90  | +0.76  | +0.40  |
| Zinc, amalgamated, . . . . .     | +0.93   | +0.82  | +0.70  | +0.44  |
| Zinc, commercial, . . . . .      | Not determ.   | +0.77  | +0.59  | +0.39  |
| Copper, . . . . .                | +0.81   | +0.62  | +0.37  | +0.16  |
| Cadmium, . . . . .               | +0.61   | +0.57  | +0.35  | ...  |
| Cadmium, amalgamated, . . . . .  | +0.55   | +0.31  | +0.19  | ...  |
| Tin, . . . . .                   | +0.45   | +0.24  | +0.17  | +0.06  |
| Bornite, . . . . .               | +0.45   | +0.25  | -0.16  | ...  |
| Copper, amalgamated, . . . . .   | +0.39 (?)   | +0.41  | -0.14 (?)  | -0.12 (?)  |
| Gold, . . . . .                  | +0.37   | +0.23  | +0.09  | -0.38  |
| Silver, . . . . .                | +0.33   | +0.15  | -0.05  | -0.36  |
| Copper-Glance, . . . . .         | +0.29 (?)   | +0.25  | +0.05  | -0.44  |
| Lead, . . . . .                  | +0.13   | +0.05  | +0.01  | ...  |
| Tin, amalgamated, . . . . .      | Not determ.   | +0.01  | -0.07  | -0.12  |
| Lead, amalgamated, . . . . .     | Not determ.   | ...  | -0.03  | ...  |
| Quicksilver, . . . . .           | -0.09   | +0.01  | -0.11  | ...  |
| Gold, amalgamated, . . . . .     | ...   | ...  | -0.13  | -0.26  |
| Antimony, . . . . .              | +0.06   | +0.03  | -0.03  | ...  |
| Arsenic, . . . . .               | +0.04   | -0.05  | -0.21  | ...  |
| Bismuth, . . . . .               | +0.00   | -0.06  | -0.20  | ...  |
| Niccolite, . . . . .             | -0.11   | -0.17  | -0.44  | ...  |
| Iron, . . . . .                  | -0.17   | -0.24  | -0.24  | ...  |
| Chalcopyrite, . . . . .          | -0.20   | -0.34  | -0.44  | ...  |
| Pyrite, . . . . .                | -0.28   | -0.42  | -0.48  | ...  |
| Galena, . . . . .                | -0.28   | -0.48  | -0.52  | ...  |
| Argentite, . . . . .             | -0.28   | -0.56  | -0.55 (?)  | ...  |
| Berthierite, . . . . .           | -0.30   | -0.52  | -0.52  | ...  |
| Speisscobalt, . . . . .          | -0.30   | -0.33  | -0.50  | ...  |
| Magnetopyrite, . . . . .         | -0.30   | -0.40  | -0.54  | ...  |
| Fahlors, . . . . .               | -0.36   | -0.52  | -0.52  | ...  |
| Arsenopyrite, . . . . .          | -0.40   | -0.45  | -0.54  | ...  |
| Platinum, . . . . .              | -0.40   | -0.46  | -0.50  | ...  |
| Cuprite, . . . . .               | -0.43   | -0.55  | -0.57  | ...  |
| Electric-light carbon, . . . . . | -0.46   | -0.52 (?)  | -0.57  | ...  |
| Blende, . . . . .                | -0.48   | -0.52  | -0.55  | ...  |
| Boulangerite, . . . . .          | -0.50   | -0.55  | -0.55  | ...  |
| Bournonite, . . . . .            | -0.50   | -0.55  | -0.56  | ...  |
| Coke, . . . . .                  | -0.52   | -0.52  | -0.42 (?)  | ...  |
| Ruby silver ore, . . . . .       | -0.54   | -0.53 (?)  | -0.54  | ...  |
| Stephanite, . . . . .            | -0.54   | -0.55  | -0.52  | ...  |
| Stibnite, . . . . .              | -0.56   | -0.56  | -0.56  | ...  |

In case (a) the zinc-copper couple would at first favour a deposit of gold on the copper, but when an appreciable quantity of gold had been deposited thereon, a copper-gold couple would be formed, in addition to the usual zinc-

gold couple. The copper would then become an anode as well as the zinc, and dissolve, though to a less degree.

On examining von Oettingen's and Christy's tables it may be observed that the weaker the solution the lower is the potential for each metal. Thus, gold has a plus value for strong solutions and a minus value for weak solutions. From this it may be inferred that, with two solutions, each containing the same percentage of gold, but one strong in KCN and the other weak, it would be more difficult to precipitate the gold from the strong solution than from the weak one. This is noticeable when the current is applied from an external source; but when dependence is placed on generating the current by immersing zinc in the solution, the results are not so apparent. This is due partly to a fall in the potential of the zinc in weak solutions, as seen in the tables, but mainly to the physical condition of the deposited metal, a matter which will be dealt with in a subsequent chapter.

A further examination of the figures will show that in the 0.65 per cent. solution the zinc-gold couple has a lower e.m.f. than the zinc-silver couple; in the 0.065 per cent. solution it is slightly higher; and in the 0.0065 per cent. solution it is considerably higher.

These electro-chemical principles lead to the conclusion that, in treating ores containing both gold and silver with strong solution, the silver precipitates more rapidly than the gold, whilst with weaker solutions the reverse is the case. This view is confirmed by results in practice.

## CHAPTER XV.

### THEORY OF THE PRECIPITATION OF GOLD AND SILVER.

#### *Section II.*

**Electro-chemical Changes.**—When simple salts of gold and silver are dissolved in water their molecules become broken down, either wholly or partly, into ions. Thus the molecules of  $\text{AuCl}_3$  dissociate into  $\text{Au}$  and  $3\text{Cl}$ , and these are free to move in all directions. If, now, an electric current is caused to pass through this solution, it directs the ions: the chlorine going to the anode (where the current enters the solution) and the gold to the cathode (where the current leaves the solution).

When, however, complex salts of gold and silver, such as double cyanides, are dissolved in water, their molecules are also broken down, either wholly or partly, but in this case some complex ions are formed, of which the gold or silver is part. Daniell and Miller<sup>1</sup> investigated the nature of the ions, and found that when an electric current is passed through a solution of  $\text{KAu}(\text{CN})_2$ , or  $\text{KAg}(\text{CN})_2$ , the  $\text{K}$  is the positive ion and goes to the cathode, whereas the negative ion consists of gold or silver and cyanogen, and this goes to the anode. Hittorf has since confirmed this, and assumes the negative ion to be  $\text{Au}(\text{CN})_2$  for gold and  $\text{Ag}(\text{CN})_2$  for silver. That is to say, when these complex salts are electrolysed, the gold and silver move in the opposite direction to that taken in the case of the simple salts.

**Electrolysis.**—Faraday discovered early last century that all ions carry the same amount of electricity, or a simple multiple of this quantity. Thus a univalent ion carries unit quantity, a bivalent ion twice that quantity, a trivalent three times, and so on. The combining power of ions is found to be conditioned by the number of units of electricity they carry. Those which carry one unit have unit-combining power, and those which carry two or three units have twice or thrice the unit-combining power.

Metals separate from solutions in proportion to their chemical equivalents or combining weights. The actual weight of each metal that separates per unit of current has been determined with great accuracy, and these are known as the "electro-chemical equivalents" of the metals. They are the quantities of metal in grams that precipitate when a current of one ampere flows for one second. Table XX. gives a few of the more important metals, with their valency and electro-chemical equivalents in cyanide solutions.

<sup>1</sup> *Phil. Trans.*, 1844, p. 1.



To determine the maximum weight of metal capable of being deposited from a cyanide solution—Let

A = current in amperes,  
T = the time in seconds,  
x = the electro-chemical equivalent,  
W = weight precipitated,

then

$W = ATx$  in grams,

or

$W = \frac{ATx}{31}$  in ounces, troy.

The above gives the maximum quantity the current is capable of precipitating, which is only obtainable with specially constructed apparatus, and from concentrated solutions of the salts. When the solution is very dilute, as in the cyanide process, the efficiency of the current falls off to a mere fraction

TABLE XX.—*Value of x in Grams per Ampere per Second.*

| Metal.           | Valency.        | Electro-chemical<br>Equivalents. | Metal.         | Valency.        | Electro-chemical<br>Equivalents. |
|------------------|-----------------|----------------------------------|----------------|-----------------|----------------------------------|
| Hydrogen, . . .  | H <sup>1</sup>  | ·000010384                       | Silver, . . .  | Ag <sup>1</sup> | ·00111800                        |
| Potassium, . . . | K <sup>1</sup>  | ·00040539                        | Copper, . . .  | Cu <sup>2</sup> | ·00032709                        |
| Sodium, . . .    | Na <sup>1</sup> | ·00023873                        | Mercury, . . . | Hg <sup>2</sup> | ·00103740                        |
| Gold, . . .      | Au <sup>1</sup> | ·00208783                        | Zinc, . . .    | Zn <sup>2</sup> | ·00033696                        |

of this quantity. However, when the number of amperes of current used are known, the time, and the actual weight of metal precipitated, it is possible to compare the efficiency of the current by the above formula for any particular case.

**Electrolysis of Simple Salts.**—When simple salts, such as AuCl<sub>3</sub>, are decomposed by the current, the Au ions give up their positive charges at the cathode and separate as atoms of metal, and the 3Cl ions give up their negative charges and become electrically neutral, i.e. form free chlorine gas. Part of the free chlorine may escape to the air, but part will decompose water at the anode, forming hydrochloric acid and oxygen.

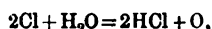
If the salt were of a highly oxidisable metal, as potassium or sodium, the K or Na ions would give up their positive charges to the cathode, just as gold does, and separate as atoms or molecules. Now, the potassium, in contact with the cathode, has a very high solution pressure in presence of water, and dissolves as if it were a local anode, whereas its cathode is the same as that of the main current. The result is that the H ions of the water give up their positive charges at the cathode, and the OH ions give up their negative charges to the potassium, or take up positive charges from the potassium, and become electrically-neutral KOH. The electrically-neutral hydrogen deposited at the cathode may become occluded to the point of saturation, or in other words

the cathode becomes polarised ; and so long as the electro-motive force of the main current is kept greater than the opposing e.m.f., due to polarisation, so long will the potassium continue to separate and decompose water. Consequently the hydrogen will continue to deposit ; and, as it cannot be occluded beyond a certain point with a given e.m.f., it escapes as gas. The reaction may be expressed in its simplest form thus—



It must not be forgotten that there is also a primary decomposition of the water that liberates hydrogen at the cathode, and oxygen at the anode, independent of the secondary action of the potassium. This occurs even with a minute current.

The anion of the salt, whose nature depends on the acid, gives up its negative charge at the anode and becomes electrically neutral. It is there left free in the solution and decomposes the water, liberating oxygen, or may combine with a salt in the neighbourhood of the anode. If the salt, being electrolysed, is KCl, the liberated chlorine attacks water thus—



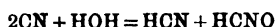
and when KOH is present



**Electrolysis of Complex Salts.**—It has been pointed out that when a complex salt such as  $KAu(CN)_2$  is electrolysed, the K ion goes to the cathode and the  $Au(CN)_2$  ion to the anode. It may be asked, if this is the case, how is it that the gold is ever deposited at the cathode? The only way the ion  $Au(CN)_2$  can get to the cathode is by a movement of the solution in an opposite direction, which overcomes the force of the current that tends to draw the gold-bearing ion away to the anode ; but it is assumed by some authorities that as the osmotic pressure of the gold is considerable, the ion breaks down again into Au and 2CN ions, and the metal then deposits as in the case of a simple salt. It is probable that this accounts for a portion of the metal precipitated, particularly when the current is of high e.m.f.

Hittorf attributes the deposition to secondary action caused by the potassium set free at the cathode. Julian and Smart considered this, and from further investigation came to the conclusion that the complex ion  $Au(CN)_2$  gives up its negative charge at the anode, becoming electrically neutral, and separates as gold and cyanogen atoms or molecules at that electrode.

By using an insoluble anode it is possible to investigate the nature of the solution collecting around that electrode without introducing metallic ions. It is found that the cyanogen liberated attacks the water and free alkali, the changes being for the most part—



and



The gold actually deposits on the anode ; but, in presence of CN ions of KCN in solution, AuCN forms at that electrode, and this compound dissolves in excess of the KCN solution to form  $\text{KAu}(\text{CN})_2$  just as if a gold anode were being used.

At the same moment that the  $\text{Au}(\text{CN})_2$  ion gives up its negative charge at the anode, the K ion gives up its positive charge at the cathode, and separates as an atom. The potassium is thereby made the positive electrode of a galvanic cell, its negative electrode being the cathode of the main current. The action of the potassium is then the same as that described under electrolysis of simple salts, i.e. local currents are continuously kept up by the separating atoms. At the same time, molecules of  $\text{KAu}(\text{CN})_2$  are carried, by diffusion of the liquid, into the paths of these local currents, whereby the ions are directed, the K cations giving up their charges at the cathode, and the  $\text{Au}(\text{CN})_2$  anions giving up their charges at the local anode ; that is, at the potassium atom separated by the main current. The gold then becomes part of the cathode of the main current, whereas the liberated cyanogen is left free to combine with the water or alkali in the vicinity.

A loss in cyanide always occurs when  $\text{KAu}(\text{CN})_2$  is electrolysed, both at the anode and cathode.

It is not at all necessary that the  $\text{Au}(\text{CN})_2$  ions should come in contact with the anode of the main current, for the reason that the useful work done by the main current is in separating potassium atoms at the cathode. The secondary action caused by these atoms at the cathode, then, does the useful work in precipitating the gold, and it is only necessary that the molecules or ions containing the gold should strike the cathode of the main current where the potassium is being separated. The potassium is not necessarily derived from the  $\text{KAu}(\text{CN})_2$  salt, but may be, and is in practice, derived chiefly from other salts in the solution. It is, of course, to be understood that other positive metals than potassium act as the anode of the secondary currents, and it would appear that under certain conditions hydrogen was capable of taking the part of a positive metal for this purpose.

**Diffusion at the Cathode.**—From the last paragraph it is obvious that one of the essential points in precipitating the gold and silver from cyanide solutions, is that of rapid diffusion at the cathode of the main current.

Working solutions in the cyanide process often contain less than 1 part of gold by weight to 100,000 parts of solution, or about 5 dwts. per ton, which shows what an enormous amount of solution has to come in contact with the cathode in order to yield 1 part of gold. It is usual, in extracting the gold, to pass the solution between several sets of electrodes until the gold contents are reduced to 10 grains per ton, or something less than one in a million. This fact alone will emphasise the importance of diffusion.

The action of the main current, as already stated, is to take the gold to its anode, and if it were not that other forces are at work, but little gold would be precipitated at the cathode. The rate at which metallic ions move by the action of the current is fortunately not great, being under 1 in. per hour with

a potential gradient of 2.5 volts per inch, and this is usually overcome to some extent by diffusion of the liquid. In practice, the necessary diffusion is accelerated by allowing the solution to flow between the electrodes at a suitable rate.

The time required for the deposition of the gold to take place when the  $\text{Au}(\text{CN})_2$  ions come in contact with the potassium anode may be considered, for practical purposes, as *nil*; and the whole of the time of precipitation is therefore occupied in bringing these ions in contact with the electrode. The time required to exhaust a solution of its metal may therefore be decreased, by either increasing the rate of flow or by increasing the cathode surface, in such a manner that a larger proportion of molecules strike that electrode. These are practical considerations which will be dealt with later.

If it were possible to remove the solution to another vessel immediately after contact with the cathode, making room for fresh solution to be acted on, perfect precipitation might result. In practice, however, the solution from which the metal had been precipitated is allowed to diffuse to the other solution still containing gold, and the result is a poorer gold solution, to be acted on in turn. Now, the poorer the solution becomes, the further apart must be the molecules or ions carrying the gold; and consequently, with a constant rate of flow, the number of ions that strike the same area of cathode surface must be an ever-decreasing quantity.

**Electrolysis by a Galvanic Couple.**—When a positive metal, having a difference of potential at two parts of its surface, is immersed in a salt of a less positive one, the dissolving part becomes the anode and the less soluble part the cathode, and precipitation of the less positive metal proceeds as if the current was applied from an external source. If the salt is a complex one, as  $\text{KAu}(\text{CN})_2$ , the action may be said to take place as follows:

It is assumed that the precipitating metal is zinc, which contains lead as an impurity; and the solution, potassium cyanide containing some  $\text{KAu}(\text{CN})_2$ . Currents are generated by the dissolving action of the zinc, as already explained, flowing through the solution from the zinc to the lead. In the paths of these currents, molecules of  $\text{KAu}(\text{CN})_2$  are carried by diffusion, and their ions are directed, the  $\text{Au}(\text{CN})_2$  going to the zinc and the K to the lead, where they simultaneously give up their charges and become atoms or molecules. The gold deposits on the part of the zinc where the current enters; that is, the anode and the  $(\text{CN})_2$  is left free to combine with KOH or water, as already stated. The K ion gives up its charge in the same way at the lead and becomes an atom, forming for the moment an anode, the lead being the cathode. Gold is then deposited on the lead as the solution diffuses to that electrode, in the way described under electrolysis of complex salts. Thus, when a zinc-lead couple is employed, gold deposits both at the zinc and the lead, whereas when a current is applied from an external source, gold is deposited permanently only at the cathode. At the moment the atom of gold is deposited on the zinc its tendency is to redissolve, but a new positive or anode part of the zinc takes this atom as a negative electrode

or cathode, on which to deposit potassium atoms. The potassium, as it separates, sustains local currents that precipitate further quantities of gold as the solution diffuses, and thus the deposit thickens. The proportion of gold actually precipitated by the primary action of the zinc must necessarily be small, as the surface soon becomes covered with gold, and the chief amount must be obtained by the secondary action of the potassium.

It is not necessary that the zinc should come in contact with the gold solution, provided that the electro-negative surface in electrical contact with the zinc is supplied, on which the potassium atoms may separate. Thus a surface of lead, tin, or iron may be employed in contact with the gold solution, the zinc being in a separate cyanide solution containing no gold, both separated from each other by a porous partition. All that is required, then, to deposit the gold is to connect the zinc with the negative surface by a wire, and so cause the gold-bearing solution to diffuse.

Neither is it necessary that the zinc should dissolve in a cyanide solution. Any other solution by which zinc is attacked will answer to a greater or less degree. In practice the solutions usually contain free alkali with the cyanide, in which case the free alkali answers the same purpose as the cyanide: that is, to produce electric currents and to deposit the alkali metal at the negative surface.

What has been said with reference to diffusion of the solution at the cathode when an external current is applied, is of equal importance when precipitating with a galvanic couple.

**Polarisation.**—If two pieces of metal, such as lead, are placed in a salt solution, such as cyanide, and are connected with the terminals of a galvanometer and the terminals of a galvanic element, it may be observed that at the moment of contact a considerable current is indicated. Later, the needle goes back almost to zero, thus showing that the current nearly stops. If, now, the element be cut out of the circuit, the galvanometer indicates a current almost equal to that of the maximum primary current, but in an opposite direction. This is known as the "polarisation current," and, like the primary current, will be found to fall in intensity, at first rapidly and afterwards more slowly. Polarisation may be due to several causes, but the chief one is owing to the accumulation of hydrogen at the negative electrode.

When a current is passed through an aqueous solution, hydrogen ions are always un-ionised at the cathode, that is, they separate in molecular or gaseous form, which up to a certain point, dependent on various factors, such as atmospheric pressure and temperature, accumulate and become occluded. Oxygen separates at the same time at the anode, and there becomes occluded. If the primary current is cut out, a hydrogen-oxygen gas cell is formed, short-circuited through the galvanometer, which results in a current in the opposite direction to that of the primary one. This current has an electro-motive force, dependent on the concentration of the gases, which, however, is never greater than that of the primary cell.

It was found in one case that when the concentration of hydrogen was at

a maximum, for a normal atmospheric pressure and a temperature of 50° F., the e.m.f. of the polarisation current was 1.1 volts. If, now, this current be opposed by connecting the electrodes with a primary cell of 1.1 volts equilibrium results and nothing happens; but if the primary cell current be less than 1.1 volts, say 0.6 volt, a polarisation current of 0.5 volt (1.1-0.6) is produced, which gradually diminishes until the concentration of the hydrogen is such as would produce an e.m.f. of 0.6 volt. On the other hand, if the e.m.f. of the primary cell is greater than 1.1 volts, say 1.3 volts, then water is decomposed, with a current having an effective e.m.f. of 0.2 volt, and hydrogen and oxygen escape to the air. It must, therefore, be remembered that this opposing force is always present when precipitating metals from solutions. By raising or lowering the atmospheric pressure, the e.m.f. of the polarisation current is raised or lowered accordingly; and by increasing the e.m.f. of the primary current, the e.m.f. of the polarisation current is also increased, but not in proportion, there being always a diminution in the latter as the former becomes higher.

## CHAPTER XVI.

### THEORY OF THE PRECIPITATION OF GOLD AND SILVER.

#### *Section III.*

**The Significance of E.M.F. for Precipitating.**—It is well known to analysts that, in the precipitation of metals by electrolytic methods, every metallic salt is only capable of being decomposed and the metal precipitated continuously when the e.m.f. current has reached a certain voltage. This varies with different metals and with different salts of the same metal. Thus the voltage for a continuous precipitation of silver from normal solutions of silver nitrate has been found to be 0·7,<sup>1</sup> whereas if a current of the same strength were applied to a normal solution of potassium silver cyanide little or no silver would be found on the electrode; and again, a current with a pressure that would just precipitate silver from a cyanide solution would not be sufficiently high to precipitate zinc from the same solution.

Le Blanc investigated this subject, and found that the cause of a metal not being continuously precipitated until a definite voltage is reached may be understood from the fact that when the metal in question is precipitated at the cathode it exhibits an opposing current, at first small, which goes on increasing to a maximum, i.e. until the deposit is in a massive form. When the pressure of the primary current is at or near that of the opposing one of the massive metal at the cathode, the minimum voltage of decomposition results, any higher e.m.f. effecting a continuous deposit. Every metal dissolved in a solution has a certain tendency to change into the metallic state, some more than others, just as every metal has a tendency to change into the ionic state, that is, to dissolve. Gold ions have a great tendency to change into the metallic state as compared to zinc ions; and, when in the metallic state, the gold has a much smaller tendency to change into the ionic state than zinc. From this it follows that, when precipitating these metals, the opposing current of the deposited metal must be greater in the case of zinc than gold, and therefore the minimum current of decomposition must be higher in the case of zinc than of gold.

When a piece of zinc is placed in a cyanide solution containing a small amount of gold, thin films or a number of specks of gold are first deposited. The difference of potential between the gold and the zinc is then of great magnitude, but this decreases as the deposit increases in thickness, until the latter assumes the massive condition of a plate of gold. For the same reason,

<sup>1</sup> Le Blanc, *Zeit. physik. Chem.*, viii. 299 (1891).

highly oxidisable metals separate as atoms with a current of low e.m.f., but they cannot assume the massive form until the e.m.f. of the primary current is greater than that produced by the massive metal.

The nature of the electrode on which the gold is precipitated has influence on the effective e.m.f. of the primary current. Thus the e.m.f. that would just precipitate gold on a gold cathode would not be effective in precipitating gold on a more highly negative substance, such as carbon, for the reason that the precipitated gold in contact with the carbon redissolves in presence of free cyanide, due to the formation of a gold-carbon couple.

**Physical Effect of the Nature of the Electrodes.**—The minimum e.m.f. of decomposition is found to depend not only on the metal and the nature of the salt, but on the physical condition of the surface of the electrodes. Thus, if the electrodes have a hard, smooth surface, such as rolled metal, the decomposition point is found to be much higher than when the surface is of a granular, spongy, or slimy nature. This is marked in the case of platinum electrodes, and is more or less marked in the case of other metals. Thus, when the electrodes consisted of hard rolled platinum foil, Julian and Smart found, at normal atmospheric pressure and 50° F., the minimum decomposition point of water to be 1.8 volts; whereas, when the electrodes were platinised, under similar conditions, the decomposition point was only 1.1 volts. When the cathode consisted of hard rolled lead, it was found that the decomposition point was always higher than when it consisted of spongy lead. Again, if a zinc plate is the positive electrode of a simple cell, rolled lead the negative electrode and the electrolyte a solution of pure  $\text{KAu}(\text{CN})_2$ , little or no gold is precipitated. If a negative electrode of spongy lead is substituted, the gold rapidly precipitates. In the first case, with rolled lead, the decomposition point was higher than the e.m.f. between the lead and zinc was capable of effecting, but in the case of spongy lead the decomposition point was reached although the e.m.f. was the same as in the first case. If the lead is jagged or granular the effect is the same as with spongy lead, differing only in degree. When lead is deposited in a spongy form on zinc, the gold precipitates for the same reason.

The phenomenon is traced to the occlusion of hydrogen. With smooth-surface cathodes the hydrogen is only slightly occluded, and a film of the gas separates the metal from the solution. The film of gas introduces a resistance into the circuit, and to overcome this a higher e.m.f. than is available would be necessary. When, however, the surface is granular or spongy, it is capable of occluding the hydrogen, and the surface film does not appear, or only partly, so that the available e.m.f. is sufficient to effect decomposition.

It is well known to those accustomed to precipitating gold with zinc shavings, that when dealing with dilute solutions the gold is apt to adhere as a thin hard film, after which the precipitating properties of the zinc seem to be almost at an end. This appears to be due simply to the raising of the decomposition point above that which the zinc is capable of effecting. The hard film of gold has the same properties as that of hard rolled metal. Anything that



will cause the surface to be roughened will lower the decomposition point. In practice, this is done by adding cyanide or alkali to the solution, so as to temporarily increase the strength of the solution. The precipitated gold is then caused to partly scale or loosen, and form a number of jagged edges, which admit hydrogen to become occluded. The addition of cyanide or alkali not only lowers the decomposition point, but slightly increases the e.m.f. between the zinc and gold, and thereby increases the current density for the time being. This causes a slimy, loose deposit to form, and thus the danger of raising the decomposition point again is diminished. In practice, this danger is often obviated by the method, originally patented by J. S. MacArthur, of dipping the zinc into acetate of lead solution before putting it into the precipitating box. This causes a loose spongy lead deposit on the zinc, which acts as the negative electrode and forms the so-called 'zinc-lead couple.' Then when the gold solution is passed through the mass of zinc-lead couples it meets with a ready formed negative surface, favourable for maintaining a low decomposition point of the solution. On the other hand, with clean zinc, a considerable time must elapse before a suitable negative surface is formed, and there is always the danger of this surface forming so as to raise the decomposition point. This danger is not so great in strong solutions as in weak solutions ;

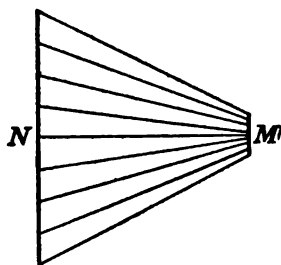


FIG. 26.—Low and high current density.

and if the solution is sufficiently strong in cyanide and rich in gold to produce a loose, slimy deposit, there can be little or no advantage in precipitating the spongy lead on the zinc.

**Influence of Current Density.**<sup>1</sup>—If the area of the anode be great and that of the cathode small, even when the current strength is low, a high-density current may result at the cathode ; but when the current is reversed, it would give a comparatively low-density current at the cathode. This is illustrated graphically in Fig. 26. If N and M represent the edges of circular discs in a solution of a salt, and a current be sent from N to M through the solution, the current becomes more dense at M than at N, as shown by the converging lines ; but if the same current be sent from M to N, the current becomes less dense at N than at M, as shown by the diverging lines. In practice it is found that with a low-density current a hard, smooth film of metal adheres firmly to the electrode, as, for instance, when a metal is precipitated at N ; but if the density is high, as at M, the deposit becomes of a granular or loose, slimy nature. These are physical effects due to current density. In precipitating a metal by a current applied from an external source, as a dynamo, the physical nature is often a matter of no consequence, but there are special cases where

<sup>1</sup> By current density is meant the ratio of the strength of the current passing, to the area of the anode or cathode. Thus, if A is the strength of the current in amperes, and B the area of the electrode in feet, then  $\frac{A}{B}$  = density of current per square foot.

it is a matter of great importance. When gold and other metals present are precipitated as a hard film, considerable difficulty has been experienced in removing the gold without destroying the cathodes. Chas. Butters, however, obviated this by employing a high-density current, which deposits the metal in such a loose form that it could be wiped off the electrode as slime or powder.

If the current comes from an internal source, as that obtained by the dissolving action of the solution on zinc, the current density is a matter of much importance. For if the current density be low, the gold precipitates as a hard film, which has the physical property of raising the decomposition point of the solution, as already explained, to such an extent that the e.m.f. between the zinc and gold is not high enough to effect precipitation.

On the other hand, if the current density be high, a loose, black, slimy deposit results, which causes the decomposition point to remain low, and then the e.m.f. of the current between the gold and the zinc is sufficient to effect continued precipitation.

Solutions in the cyanide process may contain only a small amount of gold or silver per ton, in the presence of some pounds of free cyanide per ton, that is continuously redissolving a portion of the metal as deposited. If the cathode surface is small, a given current strength may produce a thick deposit, and as only a small amount of the metal is exposed to the solution, but little is redissolved. When the cathode surface is larger, a thinner film of metal is deposited in the same time; and as it exposes a proportionally larger area to the solution, more of the precipitated metal is redissolved. An extreme case would be where the cathode was so large, for the current strength, that the metal is redissolved as fast as it is deposited.

When employing a current from an external source, it is possible to control the current strength, but when precipitating with zinc there may be, unobserved, an excessively large gold surface and a small zinc surface; consequently the current density would be low, and would allow a large portion of the precipitated metal to be redissolved.

It must not be inferred, however, that the smaller the cathode the better the precipitation, for it is obvious that the smaller the cathode the fewer will be the number of molecules that strike the surface in a given time with the same rate of diffusion. It is evident, therefore, that for every given set of conditions, there must be a certain-sized cathode that will give a maximum efficiency of precipitation. This may be determined in an empirical way when a current from an external source is used; but, when precipitating with zinc, it is necessary to take the precaution of keeping a preponderance of zinc surface exposed to the solution, and at the same time as large a negative surface as practicable.

**The Effect of Mixed Salts on Precipitation.**—Cyanide solutions that have been in use for dissolving gold and silver from ores usually contain impurities dissolved out of the ore. These to some extent accumulate, the amount of foreign substances present varying from a trace to 1 or 2 per cent.,

the total amount of KCN present being under 0.3 per cent. As might be expected, these impurities have an influence on precipitation.

When a current is passed through a mixture of salt solutions, it is generally found that the less oxidisable metals deposit with the least e.m.f., but the ions of all the salts migrate towards their respective electrodes. If the e.m.f. is increased sufficiently to deposit the more oxidisable metals, then they all deposit together, and if the more oxidisable metals are capable of decomposing a metallic salt or water, secondary action occurs which also precipitates the less oxidisable metals, when their molecules get by diffusion in contact with the cathode. If the solution contains, say,  $H_2O$ , KCN, and KCl, then the K of both salts and the H of the water go to the cathode, where, by secondary action, they precipitate the less oxidisable metals that may be in contact with the electrode at that moment, or the energy may be simply used in decomposing water. At the anode, the O, CN and Cl are separated simultaneously with the K and H. These not only act on the water, as already stated, but act on each other. It is easy to see that in mixed solutions, if a number of anions separate at the same time, they may combine with each other, or with compounds in the solution, to produce many complexes. In practice, however, the solutions are dilute, and it is improbable that the anions, as they are liberated, combine with each other to any extent. For the most part they decompose water.

When precipitating with zinc, some of the impurities introduced into the solution have a detrimental effect. The worst of these are reducing agents, such as sulphides and organic matter. Reducing agents, generally, retard precipitation of the gold and silver, by lowering the e.m.f. of the current and hindering depolarisation of the negative surface, and consequently lessen the quantity of current generated. Soluble sulphides act on the zinc, forming an insoluble coating that renders it less soluble. With strong cyanide solutions the effect of sulphides is less than with weak solutions.

Julian and Smart made some determinations of the effect of minute quantities of  $Na_2S$  in a KCN solution of 0.25 per cent. on the precipitating properties of zinc. The results are plotted in Fig. 27.

**Precipitation from Acid Solutions.**—When a solution becomes acid its dissolving action on gold and silver almost ceases, but any gold or silver that had been dissolved while in an alkaline condition still remains in solution. This is supposed to exist as hydroaurocyanic acid,  $HAu(CN)_2$ . When a current is passed through this solution between insoluble electrodes in either an acid or neutral state, the gold and silver are readily precipitated at both the anode and cathode. At the anode  $AuCN$  or  $AgCN$  is formed by the action of free HCN in the solution, and this compound retards the current when the solution is rich in metal, but in highly attenuated solutions the  $AuCN$  or  $AgCN$  dissolves in the excess of HCN. The gold or silver precipitated at the cathode remains permanent. The potential of both metals with respect to acid solutions is a minus quantity, and therefore there is little or no tendency to redissolve.

Neither zinc nor aluminium precipitate an appreciable quantity of gold from neutral  $\text{HAu(CN)}_2$  solution; but if it contains an excess of a strong acid ( $\text{H}_2\text{SO}_4$ ), precipitation is fairly rapid and perfect. For obvious reasons the use of acidified solutions is impracticable.

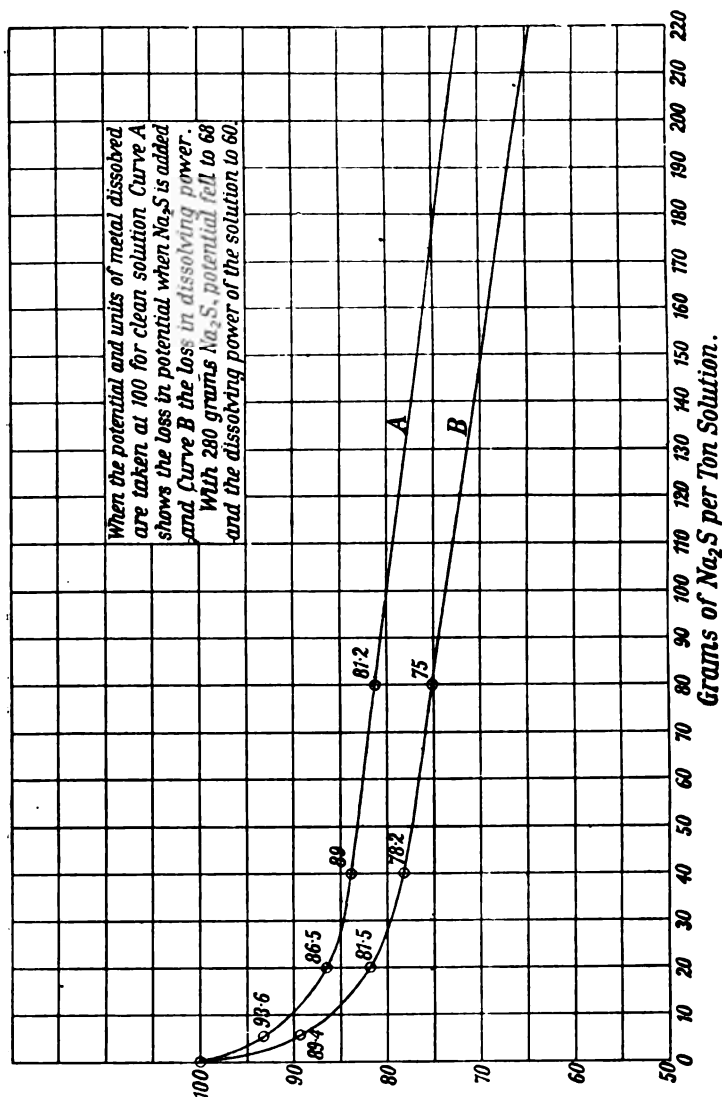


FIG. 27.—The effect of Sodium Sulphide on the precipitating properties of Zinc in a 0.25 per cent. KCN Solution.

**The Velocity or Rate of Flow past Electrodes.**—It has been pointed out that, by increasing the rate of flow through the precipitating box, a larger number of  $\text{KAu(CN)}_2$  molecules are brought in contact with the electrodes in the same time. This increase in rate of flow has also another advantage.

The cathode always becomes polarised with hydrogen, which causes the

e.m.f. of the primary current to be reduced, and if the e.m.f. of the primary current is small, it may be almost entirely stopped by the opposing e.m.f. of the deposited hydrogen. It is well known that the solution in contact with air absorbs oxygen; and if this oxygenated solution be brought into contact with the polarised cathode, depolarisation is effected and the primary current regains its original e.m.f., the reason being that the occluded hydrogen combines with the absorbed oxygen in the solution, forming  $H_2O_2$  and water.

This gain of e.m.f. is only momentary, if the solution is motionless, because the oxygen molecules in contact with the electrode become used up, and new hydrogen atoms are instantly deposited to take the place of those that combined with the oxygen. But when the solution is caused to flow past the polarised electrode, so as to produce a continued supply of oxygen molecules, a continued depolarisation results, and if the rate of flow is sufficiently rapid, the hydrogen will be removed as fast as it is deposited.

In an experimental precipitating box, specially constructed to determine the effect of the rate of flow on polarisation, it was found that when the solution had been quiescent for some hours the e.m.f. of the current between precipitated gold and zinc was 0.01 volt. The head of the box was then elevated to different heights and the solution was circulated at the greatest capacity of the box. When the slope was 1 in 30 the voltage was 0.08, when 1 in 20 it rose to 0.23, 1 in 15 to 0.33, 1 in 13 to 0.42, and 1 in 12 to 0.56, and with a greater velocity in the rate of flow the e.m.f. remained constant. The current was generated in the box by dissolving zinc in a solution of KCN and NaOH.

When the current is supplied from an external source this effect of polarisation is not a matter of great importance, as it is easy to increase the e.m.f. so as to overcome the opposing e.m.f. of polarisation. But in precipitating with zinc, the e.m.f. between the zinc and the gold, in ordinary working solutions, is seldom greater than 0.55 volt. The specific rate of flow of a solution through a zinc precipitating box is therefore a matter of practical importance.

If two boxes contain the same quantity of zinc, the column in the first having twice the length and half the sectional area of the second, then for equal volumes of solution the linear velocity of flow would be twice as great in the first as in the second. On the other hand, the solution would have twice the length of column to traverse, and therefore the time of contact with the zinc would be the same in each case. The quantity of solution passing any particle of zinc in a given time would, however, be twice as great in the first case as in the second; and, assuming a well-aerated solution, the depolarising effect in the first few compartments would be doubled. It follows, therefore, that up to the point of complete depolarisation an increase in velocity is an advantage. The sectional area of the zinc traversed should be small enough to ensure this maximum efficiency, with a length of zinc column consistent with efficient precipitation.

## CHAPTER XVII.

### THE CHEMISTRY AND PHYSICS OF SLIME AND COLLOIDS.

MUCH has been done since Graham made his classic researches on the properties of fine solid particles held in suspension in liquids. That eminent scientist distinguished and classified substances which exhibit little or no chemical affinity, and apparently dissolve in solutions without any limit, which do not crystallise, and show little or no tendency to diffuse or subside, while in many cases they are able to pass through animal membranes. He called these substances *colloids*. They are known to coagulate and settle on the addition of electrolytes and form simple substances or heterogeneous mixtures.

Modern research allows us to assume that all substances may occur in the colloidal state, but no exact definition of a colloid is possible. Graham designated colloidal solutions as *sols*, colloidal aqueous solutions as *hydrosols*. By the addition of an electrolyte the suspended substance may be coagulated as an insoluble *gel*.

*Colloid* may be roughly differentiated from *slime* by reason of the fact that the former, in its unprecipitated state, is unaffected by gravity. The most noticeable characteristic of a colloid is that a colour change invariably occurs after the addition of the electrolyte.

R. Zsigmondy<sup>1</sup> has proved that a gel may occur as a honeycomb of cells consisting of minute elastic chambers containing liquid; or as a microcrystalline structure. The same authority has shown the porous nature of silica gels by recent investigations which have indicated altered vapour pressure in the surrounding liquid.

**Artificial Preparation of Colloids.**—Crystalloidal matter may be transformed into colloidal matter by chemical or mechanical means. In 1898 Bredig showed that a bluish-red gold sol could be formed from the metal by means of an electric arc; and a very large number of elements and compounds have since been transformed from the crystalloidal into the colloidal state by various methods.

By an elaboration of Bredig's method, and by using the Poulsen arc as adopted in wireless telegraphy, R. Morris-Airey and S. H. Long<sup>2</sup> have produced colloidal solutions of gold, silver, platinum, palladium, copper, lead, iron, zinc, nickel, aluminium, magnesium, bismuth, tin, cadmium, and carbon.

<sup>1</sup> *Phys. Zeitschr.*, Nov. 15, 1913.

<sup>2</sup> *Proc. Univ. Durham Phil. Soc.*, 5, 1912-1913.

**The Particulate Theory of Colloidity.**—It is unfortunate that a haze has been allowed to envelop the question of colloidity by the confusion which followed the partial acceptance of the particulate theory, which aims to explain that colloidity is merely a state of subdivision. Experience, however, indicates that there is a marked difference between mechanically subdivided matter and colloidal matter, which it is impossible to consider is wholly due to particle size. Mechanical reduction to a slime inevitably produces a proportion of material which is infinitesimally fine. This can be separated by hydraulic methods, but cannot be made to give the true reactions of a colloid.

The pendulum has swung too far; and the diversion of attention from the particulate theory might well be explained as a needed reaction to discourage the inclusion of too many finely divided substances in the colloid category. The peculiar characteristic of a colloid is that it is colloidal, or glue-like. The originator of the term probably realised the necessity of adopting a synonym that indicated, by its very derivation, the distinction which must be recognised between colloids and non-colloidal subdivided matter.

Of late years, however, there has been an inclination to disregard the line of demarcation, and less attention has been paid to the physical peculiarities of colloids. The fluidity of water is greatly decreased by the inclusion of a true colloid, such as weathered, plastic clay. It is, however, not appreciably altered by the addition of many of the so-called colloidal metals. It would seem reasonable to infer that viscosity or fluidity should be one of the factors in determining colloidity.<sup>1</sup>

**Preferential Adsorption of Ions.**—It is assumed that, in all colloid systems, the fine particles in suspension carry statical charges of electricity; and recent research has demonstrated that these charges arise as a result of the preferential adsorption of ions from the surrounding medium.

W. C. McC. Lewis<sup>2</sup> quotes, in support of the adsorption theory of the origin of the electric charge, the results obtained by Powis in the preparation of ferric hydroxide, a compound which, as ordinarily prepared, is positively charged in an aqueous solution. It is usually obtained in this form by dialysing a solution of ferric chloride, the positive charge being attributed to the adsorption of ferric iron from the solution. By allowing a dilute solution of ferric chloride to run slowly into a dilute solution of alkali hydroxide, with constant shaking, a negatively charged colloidal ferric hydroxide can be produced, the charge in this case being attributed to the adsorption of hydroxyl ions.

**The Action of Electrolytes in Slime Settlement.**—As previously mentioned, the generally accepted view as to the reason why colloid particles, in an aqueous or other medium, remained suspended, is that they carry electric charges of the same sign. The mutual repulsion is, therefore, responsible for the maintenance of the suspension. By the addition of an electrolyte, such as lime, to a colloidal slime, a dissociation occurs, the result being that

<sup>1</sup> A. W. Allen, *Eng. and Min. Jour.*, Jan. 11, 1919.    <sup>2</sup> *Trans. Soc. Chem. Ind.*, 1919.

the metallic ions of the electrolyte are adsorbed by the slime particles. It is well known that coagulation can be effected by the addition of such a 'clarifier,' as it is sometimes termed; and it has also been proved that the sediment contains the metallic ion of the added electrolyte. It seems probable, therefore, that the charges of electricity on the particles are either neutralised by countercharges, formed during adsorption, or else the increased conductivity of the medium, as a result of the freeing of acid from the electrolyte, leads to a short-circuiting of current.

These views are, however, merely tentative; and no satisfactory or convincing explanation of the general phenomena of coagulation of colloids is available.

**The Brownian Movement.**—An English botanist, during the eighteenth century, discovered that all finely divided matter in suspension in a liquid medium is in continual movement. Any influence tending to accelerate this Brownian movement, as it is called, will tend to increase the possibility of collision between the particles themselves. Such collisions may result in union, leading to coagulation. This effect may, in certain cases, be secured by heating the solution in which the colloid is suspended. The decreased viscosity of the medium leads to an increased Brownian movement, with the result that the particles coalesce and settle.

**The Electrolytic Action of a Working Cyanide Solution.**—The double cyanide compound found in working cyanide solutions is dissociated, in the presence of water, into K (or Na) ions and Au (or Ag)  $(CN)_2$  ions. It has been pointed out<sup>1</sup> that, if the latter act as electrolytes, by which gold is deposited, the action is the result of milling in cyanide solution instead of alkaline water; and may explain why treatment after crushing in cyanide solution necessitates a longer time for the extraction of the precious metals than is required when the ore is crushed in water and, at the same time, given a preliminary alkaline wash.

It is probable that, when an ore is milled in lime water, the colloid present will assume a temporary equilibrium as far as absorption of water and salts in water is concerned; and adsorption as well as absorption will have occurred. Any further exchange of solution will then be a matter of slow dialysis. Hence, when the colloid comes in contact with cyanide solution, it will be less likely to abstract the dissolved gold and silver during the limited time of agitation and treatment.

The problem of clarification of solutions after milling in a cyanide solution is invariably obtrusive; whereas, after milling in water, trouble is seldom experienced in this direction. In other words, the electrolytic value of gold- or silver-bearing cyanide solution is inferior to that of lime water; and when once the slime particles are charged with the former, no addition of lime will materially hasten coagulation.

**Properties of Colloidal Clay.**—Kaolin is a crystalline substance which when exposed to the action of the atmosphere passes into the colloidal state.

<sup>1</sup> A. W. Allen, *Trans. Inst. Min. and Met.*, 1915.



The composition of clay is represented thus:  $\text{Al}_2\text{O}_3, 2\text{SiO}_2, 2\text{H}_2\text{O}$ . Water occurs in clay as moisture and in chemical combination.

According to Searle,<sup>1</sup> heating to  $105^\circ$  (C.) drives off the moisture, and heating to  $600^\circ$  drives off the water of combination. This point is of the utmost importance with reference to the specific gravity method of determining the dry tonnage of clayey ores. Excessive heating should be avoided so that the chemical structure of the compound is not altered. Clay is also hygroscopic, so that cooling after drying and before weighing should be carried out in a desiccator.

A theoretical knowledge of the properties and composition of clay is also necessary to the assayer. If assay samples containing such material are unnecessarily heated an alteration may take place in the compound as a result of the loss of combined water. This will mean a concentration of the ore and an erroneous result.

According to Searle,<sup>2</sup> clay has the property of selectively absorbing certain substances in solution in such a manner that they cannot be removed by washing. This point is of considerable interest in the cyanidation of silver and gold ores, and offers a reason why the extraction from slime which has weathered is often lower than when similar material is treated direct from the battery. It also throws doubt on the recognised method of differentiating between dissolved and undissolved gold and silver by washing the pulp with which the metal is associated.

In 1868 Wm. Skey<sup>3</sup> published the following results of experiments on the precipitation, by neutral salts, of clay suspended in water:

(a) A strong solution of the chloride of sodium, ammonium, calcium, magnesium or barium, or of sulphate of soda, causes an immediate aggregation of the particles, and their complete precipitation shortly afterwards.

(b) "The volume of clayey water clarified by one grain of certain of the above-named salts in 24 hours is approximately as follows":<sup>4</sup>

|                     |                             |         | Proportion. | Per cent. |
|---------------------|-----------------------------|---------|-------------|-----------|
| 1 grain common salt | clarifies 5 oz. = 1 part in | 2,187.5 | ...         | 0.046     |
| 1 " sodium sulphate | " 5 " =                     | "       | "           | 0.046     |
| 1 " barium chloride | " 10 " =                    | 4,375   | "           | 0.023     |
| 1 " lime            | " 15 " =                    | 6,562.5 | "           | 0.014     |
| 1 " sulphuric acid  | " 50 " =                    | 21,875  | "           | 0.005     |

(c) "The quantity of clayey matter present appears of secondary importance, complete precipitation having nearer relation to the degree of dilution allowed to the salt employed."

This last statement is an important one to remember in connection with slime treatment.

<sup>1</sup> *The Natural History of Clay*, by A. B. Searle.

<sup>2</sup> *Ibid.*

<sup>3</sup> "Coagulation and Precipitation of Clay by neutral Salts generally," *Chem. News*, xvii. p. 160.

<sup>4</sup> The last two columns have been compiled from Skey's figures.

In 1874 Sterry Hunt<sup>1</sup> found that the clay suspended in Mississippi water took from 10 to 14 days to settle, but that the addition of sea-water, or of salt, magnesium sulphate, alum, or sulphuric acid reduced the time required for clarification to about 12 or 18 hours.

In the same year Wm. Durham published some experimental results,<sup>2</sup> and arrived at the following conclusions :

(d) In solutions of sulphuric acid and sodium chloride the liquid cleared "in the order of the specific gravities of the solutions, so that the densest liquid settled and cleared last." This effect was more decided with acid than with salt.

(e) The suspending power of water was "*gradually increased* by the addition of small quantities of the alkalis or their carbonates, or lime. Thus water having 3 grains of sodium carbonate in it" (in about a pint) "was quite opaque for three days, while water only was seen through in a day and a half." "In sodium carbonate solutions, and most probably in all alkaline solutions," the greater part of the clay sank to the bottom, and the liquid cleared "in the inverse order of the specific gravities, so that the densest liquid settled and cleared first."

From a most interesting paper by M. J. Thoulet,<sup>3</sup> published in 1891, the following results are abstracted of a few of his many careful experiments, which were made with very fine kaolin in a graduated glass tube, placed in a d'Arsonval stove to ensure constant temperature during each settlement. The depth of clear liquid measured from the top was read off from time to time, as a measure of the subsidence.

(f) His first experiments show that in vessels of considerable size and depth, the rate of subsidence during the first part of the descent is almost constant, and that the retardation that becomes measurable later is a function of the depth, measured from the top of the still turbid portion of the liquid to the bottom of the vessel.

(g) With HCl as a precipitant Thoulet found that the rate of fall increased nearly in proportion to the quantity of acid used, a result directly contrary to the experience of Durham with H<sub>2</sub>SO<sub>4</sub>.

(h) Variations in the amount of kaolin showed that an increase in the amount of suspended matter causes a decrease in the rate of subsidence up to a certain limiting quantity, which is not stated. Beyond this point the rate is independent of the quantity of solid in suspension.

**Nature of Slime.**—Several definitions of 'slime' have been suggested from time to time, but it is not easy to find a really satisfactory one. The conception of slimes as regards their treatment by cyanide depends chiefly upon their physical properties; therefore the term 'slime' may be sufficiently defined for present purposes by saying that it includes all that part of a crushed material which, after being mixed with water and allowed to settle for forty-eight hours, retains at least 40 per cent. moisture and exhibits plastic qualities.

<sup>1</sup> *Chem. News*, xxx. p. 97.

<sup>2</sup> *Annales des Mines*, vol. xix. p. 1.

<sup>3</sup> *Ibid.*, p. 57.

The term 'slimed ore,' on the other hand, generally refers to the case where the whole of the ore is reduced to a uniform fineness, and when by far the larger proportion of the material produced passes a 200-mesh screen.

The composition of slimes varies considerably, according to the nature of the ores from which they are produced. The following analyses by Dr Lœvy,<sup>1</sup> of Johannesburg, will show the kind of material which is being successfully dealt with on the Rand. Sample A was from the Jubilee and Salisbury mine, B from the Meyer and Charlton, and C from the City and Suburban.

|   | A.    | B.    | C.    |
|---|-------|-------|-------|
| SiO <sub>2</sub> . . . .  | 72·15 | 74·51 | 69·80 |
| Al <sub>2</sub> O <sub>3</sub> . . . .                                | 16·47 | 14·08 | 17·95 |
| Fe <sub>2</sub> O <sub>3</sub> . . . .                                | 4·42  | 5·15  | 3·75  |
| MgO . . . .   | 1·26  | 1·48  | ...   |
| Ca <sub>3</sub> (P <sub>2</sub> O <sub>5</sub> ) <sub>2</sub> . . . . | 0·04  | 0·07  | ...   |
| S (total) . . . .   | 0·70  | 0·78  | ...   |
| SO <sub>2</sub> . . . .   | 0·08  | 0·02  | ...   |
| Alkalis . . . .   | 0·05  | 0·08  | ...   |
| H <sub>2</sub> O (combined) . . . .                                   | 3·40  | 2·86  | ...   |

From these figures he calculates that the probable ratio between silica and clay is 51 to 35.

**Slime Production.**—The quantity of slime produced depends upon the nature of the material crushed, the method of crushing, and the size of screen used. Talcose ores naturally give the greatest proportion of slime relatively to the amount of sand, and crystalline quartz probably the least amount. Anything which retards the delivery of the crushed ore from the crushing machine, tends to increase the proportion of slime produced. The following examples show the results obtained with several screens of different mesh when crushing Rand banket at the Langlaagte Royal mine:<sup>2</sup>

|   |      |   |   |  |  |
|---|------|---|---|--|--|
| With 1200 holes per square inch, 22·8 per cent. of slime. |      |   |   |  |  |
| „ 900 „ „ „   | 17·5 | „ | „ |  |  |
| „ 750 „ „ „   | 14·3 | „ | „ |  |  |

In 1913, with coarse crushing in mill, and regrinding in tube mills, the same kind of ore gave about 50 per cent. of slime.

**The Manner in which Settlement Occurs.**—Smart made tests on the settling of slime, of which the results are plotted in Fig. 28 to illustrate what actually occurs during the progress of the settlement, apart from any theoretical consideration.

The slime pulp in this case contained 1 part by weight of dry slime to about 2·78 parts of water. The experiments were made in a 1000-c.c. measuring glass. The pulp occupied 667 c.c.; and the upper curve *ABC* shows the depth of clear solution after each interval of 15 minutes. The figures immediately above the curve are the actual readings to the nearest whole

<sup>1</sup> *Jour. Chem. and Met. Soc. of S. Africa*, vol. ii., No. 5, p. 80.

<sup>2</sup> *S. African Min. Jour.*, April 6, 1895.

number; and the upper horizontal row of figures shows the quantity of solution cleared during the interval.

The curve shows that an appreciable time (in this case 45 minutes) is taken for the downward velocity of the slime particles to reach a maximum

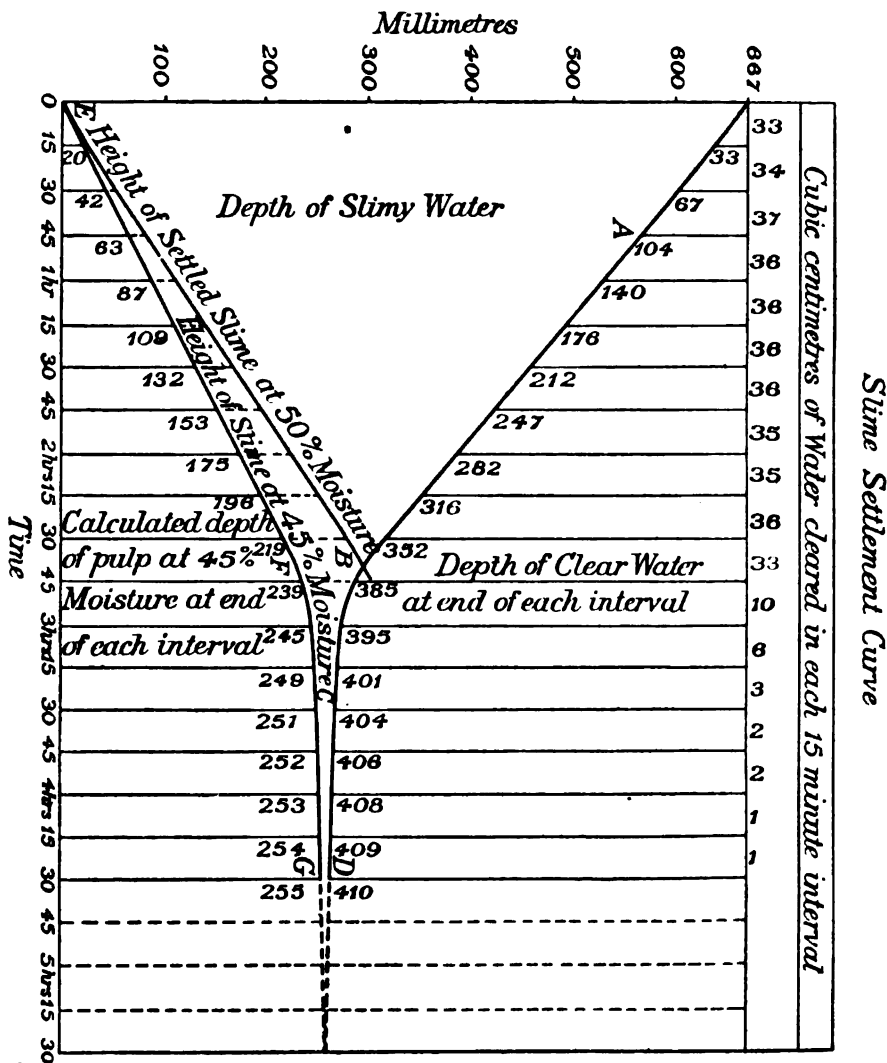


FIG. 28.

at point A. During this time the accelerating force of gravity increases the velocity until the consequent increase of resistance establishes equilibrium. The downward velocity then remains sensibly uniform to the point B, when it decreases rapidly to C. From this point to D, which marks the end of the experiment, the settlement is slow.

The obvious deduction from these facts is that during the constant velocity period the whole column of solid particles is moving steadily downwards, without thickening, until the particles themselves successively reach the bottom, where they gradually build up a column of thick slime, probably of uniform density throughout.

In order to illustrate further the action taking place, curve *EB* shows the calculated height of the column of settled slime if it settles to the ratio of 1 of solid to 1 of liquid, i.e. 50 per cent. moisture; and in the same way curve *EFG* gives the height of the thick-slime column, after each interval, if it settles to 45 per cent. moisture.

The fact that the 50 per cent. curve intersects the settling curve at point *B* indicates that the slime actually does settle to a uniform density, and that the change of velocity occurs as the last freely falling particles reach and become part of the thick column.

Therefore in a deep vat it would be expected that the settlement rate would remain practically constant until the whole of the slime particles had reached the growing layer of thick, settled slime on the bottom of the vat. This is apparently what actually occurs.

From the point *B* of the curve the remaining column of thick slime below the clear solution gradually settles still more but very slowly, as shown, to the end of the experiment. The continuation of the curves *ABD* and *EFG* in dotted lines is included to give the probable time required if the final condition of the settled slime is to be 45 per cent. moisture.

It follows from the above that, for any given slime pulp, the time required for settlement is proportional to the depth of the settling vats; and, therefore, that a given weight of slime can be settled more quickly in a shallow vat than in a deep one of the same cubic content.

R. F. Mishler,<sup>1</sup> who has published a number of experiments on slime settlement, says that the efficiency of settlement is independent of the depth, but this assumes that the vats of various depths are all of the same diameter, but when the comparison is made between vats of the same cubic capacity, the time result is in favour of the shallow vat. A considerable depth is, however, advantageous because it allows a relatively larger quantity of clear solution to be decanted. Mishler has given figures showing that the rates of settling in practice agree with the results of laboratory tests through a wide variation of dilution, namely, from 1·7 to 1 up to 8 to 1 dilution ratio.

**Use of Lime for Settlement.**—At the present time, lime may be said to hold the field at the chief gold and silver mining centres as the cheapest and most efficient settling agent. It is sometimes added to the ore before the latter reaches the mill, sometimes in the mortar boxes, and frequently between the sand-collecting vats, where these are used, and the slime plant, either as a freshly and finely ground powder or as an emulsion in water or solution.

**Secondary Uses of Lime.**—In the great majority of instances the use of

<sup>1</sup> *Eng. and Min. Jour.*, Oct. 5, 1912.

lime is imperative in the treatment of gold and silver ores by a wet process. The water or solution used for milling purposes must be freed from suspended solids. The natural slime in the ore, as well as the ore which has been slimed in the mill, must be settled in a minimum of time; and the resultant sludge must contain a minimum of moisture. By the use of lime as a coagulant, the disadvantages, both metallurgical and economic, which arise from the necessity of collecting slime in large dams, are avoided.

J. R. Williams<sup>1</sup> has pointed out that a larger percentage of gold is caught in an amalgamation plant when lime is used. This is due to the fact that the slime is coagulated by the electrolyte and a greater proportion comes into contact with the plates. This contention is supported by the figures in the following table:

| Name of Mine.          | Percentage of total Gold caught by Amalgamation. |            | Gold Content of Slimes in dwt. |            |
|------------------------|--|------------|--------------------------------|------------|
|                        | Without Lime.                                    | With Lime. | Without Lime.                  | With Lime. |
| Geldenhuis Deep, . . . | 49·63  | 54·53      | 4·21                           | 2·93       |
| Treasury, . . . .      | 55·60  | 61·50      | 4·01                           | 2·55       |
| Crown Reef, . . . .    | 50·01  | 57·81      | 6·16                           | 4·50       |
| Bonanza, . . . . .     | 55·71  | 59·81      | 8·00                           | 5·75       |

The reduction in gold content in slime which results from the use of lime and improved amalgamation is a matter of importance when the building of a slime plant is contemplated, for it is evident that the assay of the slime obtained when lime is not used is not a safe basis for the calculation of the probable results of such a plant. It is, therefore, necessary to ascertain by the experimental use of lime in the mill water, in cases where it is not already used, whether the slime, under the new conditions of improved amalgamation extraction, will be rich enough to pay for treatment.

**Use of Lime in Practice.**—At the Goldfield Consolidated Mill, Nevada, J. W. Hutchinson<sup>2</sup> found that the maximum settlement (to about 40 per cent. moisture) occurred with water containing from 0·015 per cent. to 0·025 per cent. CaO, and a further variation either way of 0·01 per cent. reduced the rate of settlement. When the CaO content reached 0·075 per cent. the tonnage settled in a given time was reduced by about one-quarter, and the greatest density obtained in the settled slime was only 1·3, corresponding to about 55 per cent. moisture. These figures confirm, on a working scale, the experimental results given above.

<sup>1</sup> *Jour. Chem. and Met. Soc. of S. Africa*, vol. ii. p. 87.

<sup>2</sup> "Lime in Cyanidation," *Eng. and Min. Jour.*, July 27, 1912.

At the Consolidated Mill the lime is first soaked, then dumped into two 4-ft. grinding pans run without mullers. These overflow a lime emulsion which is distributed to the required points by sheet-steel launders.

At El Oro Mill,<sup>1</sup> Mexico, the unslaked lime is fed by hand to the conveyor belt which carries the ore to the mill. On account of the acidity of the ore, the solutions are kept up to about 0.1 per cent. of CaO, with an average consumption of 15 lb. of lime per ton of ore.

At the West End Consolidated Mill,<sup>2</sup> Tonopah, the solution titrating 0.05 per cent. CaO gives a thickened pulp from the Dorr vats of 1.28 to 1.30 sp. gr., *i.e.* from 71 per cent. to 55 per cent. moisture.

<sup>1</sup> C. E. Rhodes, *Eng. and Min. Jour.*, xxx., Aug. 10, 1912.

<sup>2</sup> J. P. Hart and H. O. Williams, *Eng. and Min. Jour.*, July 27, 1912.

## CHAPTER XVIII.

### ADSORPTION AND ABSORPTION PHENOMENA.

ADSORPTION has been defined as the "adhesion of molecules of gases or dissolved substances to the surfaces of solid bodies, resulting in a relatively high concentration of the gas or solution at the place of contact."

**Adsorptive Properties of Charcoal.**—The adsorption of gases by charcoal was discovered during the eighteenth century, and subsequent investigations have shown that a number of substances may be concentrated in the same way. G. C. Schmidt<sup>1</sup> has published the results of investigations on the adsorptive powers of charcoal when in contact with various substances. He shows that equilibrium, or maximum adsorption, depends on the quality of the charcoal; also that if charcoal is brought into contact with a solution containing two substances, *A* and *B*, it adsorbs less of both of these substances together than it would from a simple solution of either. If maximum adsorption has been reached with a solution of *A*, and then a solution of *B* is added, the result is that the charcoal relinquishes a proportion of the *A* substance in order to take up some of the solution of *B*.

**Adsorption and the Cyanide Process.**—The adsorptive power of charcoal has a distinct bearing on the cyanidation of ores, and this substance causes a loss of metal when it is inadvertently included with the ore for treatment. The loss is undoubtedly due to adsorptive influences, either direct or indirect. In practice the adsorptive characteristics of charcoal have been utilised for the precipitation of the gold and silver from cyanide solution, and the methods adopted are described in a subsequent chapter.

Investigations have shown that other substances besides charcoal have marked adsorptive properties. Among these may be mentioned clay and kaolin. Colloidal substances would seem to possess absorptive as well as adsorptive properties.

Adsorption phenomena are developed to a considerable degree in the cyanide process, where there is an enormous area of contact-surfaces between the solids and the solution, and with media, such as silica or hydroxide of iron, highly favourable for producing the effects.

Gore<sup>2</sup> specially prepared pure silica, and in a number of cases he agitated 25 c.c. of different salt solutions of different strengths with 50 grains, in each case, of the silica. The supernatant liquid was then analysed and the

<sup>1</sup> *Zeitschr. Phys. Chem.*, 74-76, 1910.

<sup>2</sup> *Proc. Birm. Phil. Soc.*, vol. ix. part i.



results tabulated. Those in the case of KCN and NaCN are given below in Table XXI., together with the losses of cyanide in pounds per ton of silica.

TABLE XXI.

| Original Solution KCN. | Residuary Solution after contact with Silica 16-24 hours. |           | Original Solution NaCN. | Residuary Solution after contact with Silica 16-24 hours. |            |
|------------------------|---|-----------|-------------------------|---|------------|
|                        | Loss per ton Silica.                                      | KCN loss. |                         | Loss per ton Silica.                                      | NaCN loss. |
| per cent.              | lb.   | per cent. | per cent.               | lb.   | per cent.  |
| 0·025                  | 8·7   | 87·5      | 0·025                   | 3·7   | 86·7       |
| 0·25                   | 21·8  | 49·0      | 0·1                     | 2·9   | 16·6       |
| 1·0                    | 45  | 26        | 1·0                     | 26·6  | 15·4       |

The result obtained with the 0·1 per cent. NaCN solution seems to be irregular. The loss of cyanide appears excessive when compared with the low loss per ton in practical cyaniding, but Gore experimented with fine precipitated silica, which is a far more effective agent for demonstrating adsorption phenomena than the sands usually treated by the cyanide process.

It would appear from the above results that the actual loss in cyanide is greater the stronger the solution, but the percentage loss is greater the weaker the solution, and this is what is found in practice. The influence of time of contact and temperature, between the limits ordinarily employed in the cyanide process, have little or no effect on the amount of salt adsorbed. A mixture of salts as usually found in working solutions produces a disturbing influence on the amount adsorbed, which becomes irregular. Other substances mixed with the silica, as alumina or oxide of iron, may affect adsorption materially.

A freshly formed precipitate, as when an alkaline solution is added to an iron salt, so often occurring in the process, carries down with it a much larger percentage of the alkali than a similar precipitate previously prepared.

**Investigation of the Nature of Adsorbed Substances.**—Linder and Picton and others have studied this subject, and have found that the solid particles actually decompose the electrolyte, carry down and adsorb the positive ion, while the negative ion is liberated and remains free in the solution. Whitney and Ober<sup>1</sup> have extended these investigations and made accurate determinations on arsenious sulphide in colloidal state, using various salts as precipitants—barium, strontium, calcium, and potassium chlorides—and proved that the precipitate causes the salt to hydrolyse, carry down the base, and leave the negative ion free in solution. Also, the base carried down and adsorbed was in proportion to the chemical equivalent weights of the barium, strontium, calcium, and potassium.

<sup>1</sup> *Jour. Am. Chem. Soc.*, vol. xxiii., No. 11, Nov. 1901.

The same authorities give good reasons for asserting, what is doubtless true, that the fine solid particles cause hydrolysis of the salt, whereby the acid is left free in the solution, while the metal is carried down and adsorbed as a basic hydroxide.

In the case of working cyanide solutions the presence of impurities produces complications and causes irregular results in the loss by adsorption. A slight excess of caustic alkali, as found useful in practice, has a marked effect in reducing this loss, while the presence of carbonates increases it.

With slimes and sands, such as one deals with in the cyanide process, the adsorption phenomena is not developed to such a high degree as in the case of the so-called colloids, but colloidal precipitates often occur in the treatment. Little is known about the laws of adsorption, but it is generally believed that there is a proportionality between the surface-area of the solids and the amount adsorbed from the same solutions, and by solids of the same nature.

M. J. Thoulet<sup>1</sup> made the following experiment to show that solid particles carry down air as well as salts in solution. A vertical tube is filled with boiled water, to which pyrogallie acid is added with a little gum arabic to retard the oxygenation. Fine grains of solid matter are then dropped into the tube, and as they reach the bottom each one is surrounded by a dark aureole, "showing well that it has fixed and carried down an envelope of air."

**The Action of Charcoal on Aurocyanide Solutions.**—The contention that the precipitation of gold or a gold compound from aurocyanide solutions, by means of charcoal, is due to adsorption, is maintained in a monograph on the subject,<sup>2</sup> from which the following is abstracted:

Most of the research work in connection with the precipitation of gold or a gold compound by charcoal has centered around a determined effort to prove that carbon monoxide gas is the cause. The investigations of Morris Green<sup>3</sup> lent considerable weight to this theory. His conclusion that the carbon monoxide occluded in the pores of the charcoal was the active agent for the precipitation of gold from cyanide solution was based on experimental work, which went to show that charcoal from which the gases had been withdrawn, *after considerable heating*, was a poor precipitant when compared with charcoal not so treated.

A table (No. XXIA.), showing the volumes of gases abstracted under various conditions, is given herewith, and has been compiled from Green's monograph. By removing the amounts shown in Cols. 1 and 2, the ability of the charcoal to precipitate gold was practically unimpaired. In Col. 3 a test is shown when the charcoal was subjected to moderate heat and high vacuum without appreciably affecting its precipitating power. The figures given in Col. 4 indicate the volumes of gases recovered by subjecting the charcoal to a high temperature (just above the decomposition point of carbon

<sup>1</sup> *Annales des Mines*, "Experiments in Sedimentation," vol. xix. p. 1 *et seq.*

<sup>2</sup> A. W. Allen, *Trans. Inst. Min. and Met.*, 1918.

<sup>3</sup> *Trans. Inst. Min. and Met.*, vol. xxiii. pp. 65-78.

monoxide into carbon dioxide and oxygen). After this test the charcoal was cooled away from the atmosphere and afterward used as a precipitant. Tests showed that its power had been considerably reduced. The conclusion with reference to the action of carbon monoxide gas was based on a comparison of these results with the well-known fact that inert or exhausted charcoal, after heating in contact with the atmosphere, becomes revived, and its precipitating powers largely restored.

TABLE XXIA.—*Evolution of Gases from Charcoal.*

| Gas.                                | I.<br>5 gm. Freshly<br>Ignited Char-<br>coal to 20-in.<br>Vacuum,<br>c.c. | II.<br>5 gm. Exposed<br>Charcoal to<br>20-in. Vacuum,<br>c.c. | III.<br>Charcoal from<br>II. heated to<br>70° High<br>Vacuum,<br>c.c. | IV.<br>Charcoal from<br>II. heated to<br>500° High<br>Vacuum,<br>c.c. |
|-------------------------------------|---|---|---|---|
| CO <sub>2</sub> . . . .             | 0·0   | 0·16  | 1·50  | 155·00  |
| O . . . .                           | 10·76   | 18·28   | 8·58  | 8·75  |
| CO . . . .                          | 12·30   | 0·90  | 0·40  | 327·50  |
| Methane . . . .                     | 0·00  | 0·49  | 0·52  | 0·00  |
| H . . . .                           | 22·75   | 45·10   | 5·85  | 31·25   |
| N . . . .                           | 15·68   | 17·05   | 48·16   | 727·50  |
| Total volume of gases<br>expelled } | 61·49   | 81·98   | 65·1  | 1250·00   |

Green's conclusions relied on the inference that the immense volume of carbon monoxide gas, which may be obtained by heating and exhausting charcoal, existed in the same chemical composition, although in a much more condensed and active state, on the surface of the charcoal particles. The negative results obtained by a comparison of the precipitating powers of charcoal before and after abstraction of the carbon monoxide at low vacuum and normal or moderate temperatures were explained by the assumption that the "loosely held gases" were valueless in this connection, an inference which was perhaps only a repetition of the fact that carbon monoxide has no action whatever as a precipitant of gold from cyanide solution when applied experimentally.

The work of Rhead and Wheeler<sup>1</sup> on the combustion of charcoal is of pertinent application in considering the precipitation of gold from aurocyanide solution. They summarise their conclusions by stating that "in the normal burning of carbon the CO<sub>2</sub> and CO, found as the apparently primary products of combustion, arises from the decomposition, at the temperature of combustion, of a complex whose formation is the first result of the encounter of oxygen and carbon molecules. This complex, to which the hypothetical formula C<sub>2</sub>O<sub>2</sub>, has been given, shows variation in composition and cannot be isolated by physical means. When released by increase of temperature and

<sup>1</sup> *Journ. Chem. Soc.*, vol. cxiii.

*exhaustion* it appears as CO and CO<sub>2</sub>." The italics have been added to emphasise an important point.

O. C. Ralston<sup>1</sup> referred to this fact as explaining the sudden evolution of a great volume of oxides of carbon when Green<sup>2</sup> heated his charcoal *in vacuo* to 500° C., as a preliminary to the only test of this kind which resulted in appreciably decreasing the precipitating power of the charcoal. It is difficult to see how this fact can be used in support of the carbon-monoxide theory, when the gases are not formed or liberated without an increase of temperature and decrease of pressure incompatible with practical precipitation conditions with aurocyanide solution.

According to W. R. Feldtmann,<sup>3</sup> adsorption plays an insignificant rôle in the reaction, the principal result being apparently a chemical precipitation. A statement of the variation in result with solution of graduated gold content is given in Table XXIB. This is compiled from Feldtmann's paper, and it will be noticed that the third column contains figures referring to what he considered adsorbed gold. These were adduced by the following method:—"Of the 200 c.c. of solution, applied to the charcoal in each test, exactly one-half was filtered off and the gold in it determined. The remainder of the solution was then passed through the same filter and the charcoal washed with a copious supply of water. The second half of the solution (plus washes) was then assayed. . . . In each case this contained slightly more gold than the first half, showing that, by virtue of surface action, there is a small but distinct concentration of the soluble (and therefore presumably unchanged) gold salt on the charcoal. This concentration of salts in solution is an effect which, as will be noted from the above series of tests, is different in nature and of a different order from permanent precipitation. The precipitation results are based on the amount of gold originally in the 200 c.c. of solution, minus the amount contained in the whole of the solution and washes after precipitation."

This explanation infers a conception of adsorption which differs from the generally accepted view, especially when consideration is paid to the fact that adsorbed substances are usually removed from the adsorbent, by washing, with extreme difficulty. The alternative explanation that the figures in the final column refer to the gold (as aurocyanide solution) mechanically held in the charcoal by simple imbibition is strengthened by the fact that the figures do not show the definite and regular variation that might be expected in an instance of adsorption from solutions of graduated metal content.

H. R. Edmands<sup>4</sup> accepts the earlier work of Green and Feldtmann in support of the carbon-monoxide theory, but it is difficult to conceive how the practical results obtained in working operations can be used in this connection. Attention is called to the fact that water-quenched and wet-crushed charcoal is twenty times more efficient as a precipitant than the air-dried,

<sup>1</sup> *Min. and Sci. Press*, July 17, 1915.

<sup>2</sup> *Ibid.*

<sup>3</sup> *Trans. Inst. Min. and Met.*, vol. xxiv, pp. 329-43.

<sup>4</sup> *Bull. Inst. Min. and Met.*, Mar., 1918.

dry-crushed material. It follows, therefore, that a preliminary treatment tending to eliminate a great proportion of occluded gas results in accelerating the precipitating action. There seems little evidence in this fact to add weight to the carbon-monoxide theory, but rather the reverse. If it can be considered that the precipitation of the gold is not a chemical action but a physical one, then it is easy to see why the removal of occluded gas must necessarily increase the surface activity of the charcoal.

As already stated, carbon monoxide is without effect on an ordinary aurocyanide solution if applied by any known method. It is possible to prepare a gold sol, however, by bubbling the gas through a dilute solution of gold chloride; but this is a distinct case of colloid formation, with no bearing on the present problem. When such a sol is prepared from gold chloride, the chlorine is freed and hydrochloric acid found in the resulting solution. With precipitation of gold by charcoal there is no likelihood of a gold sol being formed, and there is no evidence of the release of the quantity of cyanide that would necessarily accompany such a change. A silver sol can be obtained in the same way with carbon monoxide.

Concerning the carbon-monoxide theory, it is to be noted that recent work on a practical scale undoubtedly proves that wet crushing and fine grinding are two of the fundamental necessities for effective precipitation. It would certainly seem that wet crushing for the time specified as found necessary (18 hours) would tend to eliminate more occluded gas than if dry crushing was practised. The success of the Moore-Edmands process (see chapter on "Precipitation by Aluminium, Charcoal, and Sodium Sulphide") might be ascribed to the fact that the quenching excludes the bulk of whatever gases have been held in the pores of the charcoal, and that wet grinding is a continuation of this action. A treatment that increases the surface area of the charcoal and eliminates occluded gas is one that insures maximum surface energy. The adsorption explanation is generally accepted as the most reasonable one to account for the extraordinary power of surface concentration possessed by charcoal in connection with the abstraction of gases and matter in solution; and a modified explanation would seem to be equally applicable in the case of the precipitation of a metallic compound from gold-bearing cyanide solution.

One of the most prominent characteristics of adsorption, as Philip states,<sup>1</sup> is that the amount of the dissolved substance taken up by the solid increases much more slowly than the concentration of the solution, and that the removal of the substance is relatively most complete in dilute solutions. This is not a characteristic of any other reaction, but a further reference to Table XXIB. shows that charcoal precipitation obeys this law. If this argument is carried to a conclusion it will be found that the best proof is to plot the logs of the amount adsorbed, in proportion to weight of adsorbent, against the logs of the concentrations of the solution; then the curve obtained should approximate a straight line.

<sup>1</sup> *Physical Chemistry*, by J. O. Philip.

TABLE XXIB.—*Adsorption and Precipitation of Gold by Charcoal according to Feldtmann.*

| Gold in 200 c.c.<br>Solution, Mg. | Gold Precipitated,<br>Mg. | Gold Adsorbed,<br>Mg. |
|-----------------------------------|---------------------------|-----------------------|
| 2.16                              | 2.08                      | 0.08                  |
| 5.4                               | 4.28                      | 0.02                  |
| 10.8                              | 7.4                       | 0.20                  |
| 16.2                              | 9.75                      | 0.45                  |
| 21.6                              | 11.29                     | 0.9                   |

In the case of an aqueous solution of acetic acid Schmidt gives the following figures :

| Concentration<br>of Solution. | Weight<br>Adsorbed. |
|-------------------------------|---------------------|
| 0.0365                        | 0.93                |
| 0.084                         | 1.15                |
| 0.135                         | 1.248               |
| 0.206                         | 1.43                |
| 0.350                         | 1.62                |

From the logs of these figures it is possible to plot the curve as shown in Fig. 28A, when it will be found that deposition obeys the adsorption law. By using the same argument in the case of gold precipitated from

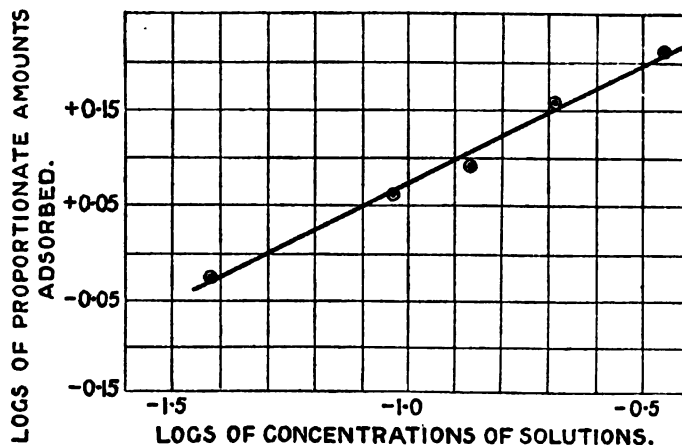


FIG. 28A.—Adsorption Curve ; Acetic Acid and Charcoal.

cyanide solution and by plotting the logs of the figures given by Feldtmann, and reproduced in Table XXIB., in the same way it is possible to obtain a precisely similar curve (Fig. 28B).

Adsorption is a surface phenomenon, so much so that it has been suggested that surface area should be estimated by adsorptive effect. This effect is directly proportionate to surface exposure, but it would be impossible

to estimate the area in the case of charcoal by calculation from screen analysis, as the larger sizes would possess a greater surface, due to porosity, than that indicated by the cube of the diameter of the average mesh measurement. If this fact is taken into consideration, the figures given in Table XXIC., and

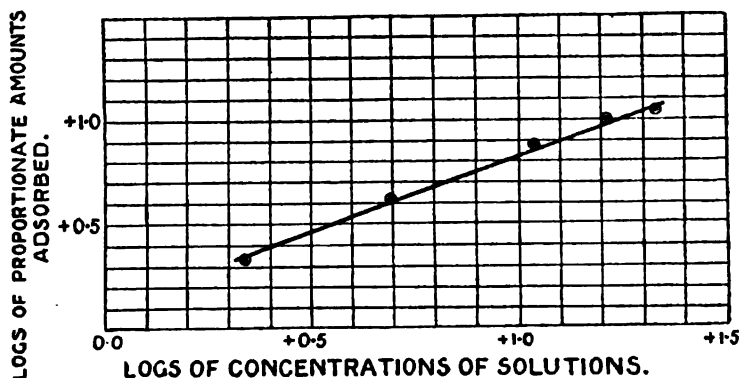


FIG. 28B.—Adsorption Curve ; Auricyanide Solution and Charcoal.

quoted by Edmands, are interesting and significant. This datum could well be used in support of the adsorption theory as an explanation of the precipitation of gold, but it is difficult to see that it could be used in support of the contention that occluded carbon monoxide gas is the cause of the action.

TABLE XXIC.—Particle Size and Efficiency of Precipitation.

| Size of Charcoal,<br>In. Mesh. | Gold Precipitated,<br>Mg. |
|--------------------------------|---------------------------|
| - 1/2 + 20                     | 0.806                     |
| - 20 + 60                      | 0.806                     |
| - 60 + 100                     | 1.211                     |
| - 100 + 150                    | 1.617                     |
| - 150                          | 3.229                     |
| - 150 reground                 | 4.442                     |

Solution from which potassium or sodium-gold cyanide has been adsorbed by charcoal should show a loss of cyanide in proportion to theoretical considerations. Edmands has proved this to be the case, although his figures were submitted in order to support the contention that a compound of the composition  $\text{KAu}(\text{CN})_2\text{CO}$  was formed. As the same amount of cyanide would be lost in each case, the results are equally applicable in support of the simple adsorption theory.

The effect of the addition of acid or alkali often furnishes evidence of the validity of certain reactions. S. B. Christy<sup>1</sup> shows that acidity in the

<sup>1</sup> *Trans. Am. Inst. Min. Eng.*, vol. xxiv.

solution results in a considerable increase in precipitating power. Edmands has since confirmed this. The data published in this connection afford further evidence in favour of the adsorption theory, and a parallel instance of the retarding effect of the addition of alkali and the accelerating effect of the addition of acid has been demonstrated by Lachs and Michaelis.<sup>1</sup> These investigators proved that the adsorption of potassium chloride in water by charcoal was diminished by the addition of KOH, and hastened by the addition of acid. This result was attributed to the obstructive effect of hydroxyl ions as compared with hydrogen ions.

A feature of adsorption phenomena is the difficulty experienced in removing the adsorbed substance by washing in excess of the solvent or water. This is particularly noticeable in the case of the removal by charcoal of a gold compound from an aurocyanide solution. Considerable attention was paid by Feldtmann to the fact that the gold precipitate on charcoal was soluble, to some extent, in alkaline sulphide solutions. Edmands has elaborated this phase of the question, and also shows that the addition of even a small quantity of soluble sulphides to the cyanide solution considerably reduces the precipitating power of the charcoal.

Adsorption is an equilibrium and closely dependent on surface-tension influences. By varying the surface tension by the addition of other reagents, the amounts adsorbed may also be varied, but an equilibrium is again reached, nevertheless. If a soluble sulphide is added to aurocyanide solution in a charcoal pulp, the equilibrium is disturbed and a new equilibrium established. It is not so much the question of the solubility of the gold in the sulphide solution as the alteration in the physical condition of the solution which causes a release of a part of the adsorbed compound. Conversely, the same effect is produced and the same equilibrium reached if the soluble sulphide be added to the cyanide solution before contact of the latter with charcoal. F. Wartenweiler<sup>2</sup> showed that strong reagents which do not dissolve the gold precipitate on charcoal do not inhibit the precipitating action when added to the solution. This is also in accord with the known laws of adsorption. Platinum adsorbs picric acid from aqueous solutions, and the adsorbed acid is insoluble in water. The addition of alcohol disturbs the adsorption equilibrium and releases the picric acid; but from this fact it would be unwise to infer that picric acid is insoluble in water and soluble in alcohol.

**Absorptive Nature of Colloidal Clay.**—The presence of clay in so many geological formations, and its invariable association with other metalliferous ores, makes the study of its properties one of considerable importance. The data following have been taken from a paper<sup>3</sup> on the influence of clay on ore-dressing and cyaniding operations.

Clays have been encountered in cyaniding operations which gave little or

<sup>1</sup> *Zeit. f. Elektrochemie*, 1911.

<sup>2</sup> *Trans. Inst. Min. and Met.*, vol. xxiv. pp. 360-363.

<sup>3</sup> A. W. Allen, *Trans. Inst. Min. and Met.*, 1916.



no trouble. In other instances the difficulty of percolation has led to the condemnation of filters of various kinds; the recovery of metal is seldom so satisfactory as might be expected from its condition of minute subdivision; classification problems are unavoidable as a result of the decrease in the fluidity of the pulp, and settlement and dewatering is slow and only feasible in comparatively highly alkaline or acid solutions. The recovery of metal from clayey material which has been allowed to accumulate for any length of time is invariably less than when treated direct. The customary explanation assumes that the associated oxygen is absorbed during weathering by the decomposition of pyritic and other material in the clay, resulting in a decreased solution efficiency when the slime is brought into contact with cyanide.

It is now known, thanks to contemporary scientific investigation, that there is a fairly sharp line of demarcation between colloidal and non-colloidal clay. The two classes may be found together in all proportions, but their characteristics are, nevertheless, distinct.

The weathering of clay produces a marked effect on the physical properties of the compound. Rohland<sup>1</sup> suggests that this may be due to micro-organisms. On the other hand, it would be idle to speculate on the precise chemical changes which may occur under such conditions. Suffice to say that the original clay is practically non-absorbent, whereas the weathered material is highly absorbent. In its non-colloidal state it is completely unplastic, whereas after decomposition it becomes highly plastic and capable of intricate manipulation.

The absorptive powers of clay are by no means confined to the abstraction of water; and Zsigmondy<sup>2</sup> has pointed out that the absorption by colloid material of this description may be so marked that the products resulting may be mistaken for new chemical compounds. The physical difference between the absorptive powers of colloidal as compared with non-colloidal clay has been placed on record by a number of investigators; and Bourry has shown that whereas kaolin may not absorb more than 2 per cent. of a salt from a solution, a plastic clay may take up from 10 to 20 per cent. of the same substance under similar conditions. It follows, therefore, that the passage of a substance from the non-colloidal to the colloidal state is accompanied by marked change.

Van Bemmelen (quoted by Foote<sup>3</sup>) has shown that the colloidal precipitates of aluminium hydroxide are not compounds of definite composition, and he considers that the amount of absorption is dependent on the structure of the jelly. Hatschek<sup>4</sup> deals at length with the absorptive powers of gels, and refers to Graham, who, in his preliminary work, noticed that substances diffused into and out of such compounds. Zsigmondy<sup>5</sup> has proved the

<sup>1</sup> *Bio-Chem. Zeitschr.*, 1912.

<sup>2</sup> *Colloids and the Ultra-Microscope.*

<sup>3</sup> *Jour. Amer. Chem. Soc.*, 1914.

<sup>4</sup> *An Introduction to the Chemistry and Physics of Colloids.*

<sup>5</sup> *Phys. Zeitschr.*, Nov., 1913.

absorptive nature of silica gels by noting the altered vapour pressure of the surrounding solution. Hirsch,<sup>1</sup> Rohland,<sup>2</sup> and other investigators, have shown that the higher the concentration of the solution the greater the quantity of the salt absorbed by the clay. From this evidence it would seem that the common action with emulsoid clay is one of absorption.

The displacement of retained bases in clay, by washing, would seem to be a lengthy process under natural conditions. Bourry gives instances where beds of ferruginous clay have been deprived of the whole or part of the iron compounds they contain, after having been washed for centuries with water containing organic acids in solution.

In wet processes of ore reduction, and when clay compounds are milled in a cyanide solution containing gold, it is probable that an absorption of such solution occurs in the honeycomb cells of the colloid; and that subsequent displacement of the metal must necessarily occupy a much longer time than did its absorption. The absorption is due to direct imbibition, but displacement under normal conditions is due to a slow-acting osmotic pressure when the colloid slime is surrounded with wash solution or water. It is also probable that any absorptive equilibrium is only temporary under such conditions, and that cell content varies with the composition of the surrounding solution.

The question arises as to the merits of a slow decantation, as compared with a rapid filtration process for the removal of the dissolved metal. It would seem that a combination of the two, in the order stated, would be the most efficient.

It is generally recognised among metallurgists that a considerable time of contact is necessary in washing operations with certain classes of ores; and that theoretical displacement figures do not hold good in practice. From what has already been said, it is apparent that time of wash should not always be based on estimates of gold in solution which is apparently dissolved, as compared with gold associated with the ore which is apparently undissolved. In practical work it is generally found that, although the metal which has been dissolved may be, apparently, replaced fairly rapidly by ordinary means, a further abstraction of metal from the ore occurs after prolonged contact with wash solution. This fact furnishes additional evidence that the gold solution so recovered is, in part at least, being drawn from the cells of the colloid slime, and not merely washed from the surface of the slime particles.

On some of the Kalgoorlie mines the value of prolonged washing is not so obvious, and comparatively short washes are given which abstract high percentages of recoverable gold from the ore. In this instance it is probable that previous roasting treatment has, in some cases, destroyed the colloidal envelope, thus reducing the possibility of absorption to a minimum.

In order to verify the contention as to the ill-effects of colloidal interference, a number of experiments were carried out. A sample of colloidal slime, free from sulphides and organic matter, and carrying neither gold nor

<sup>1</sup> *Thonindustrie-Zeitung*, 1904.

<sup>2</sup> *Bio-Chem. Zeitschr.*, 1909.

silver, was obtained. This was finely pulverised and agitated for six hours with an alkaline cyanide solution of normal cyanide content, and carrying 6.6 dwt. of gold per ton. At the end of this time the pulp was divided into two parts, one portion was drained of surplus solution until the slime carried a moisture percentage of 75, carefully dried, and assayed entire. The result of the assay was 8.2 dwt. per ton, which showed an adsorption and absorption of, approximately, 3.25 dwt. per ton.

The other portion was thoroughly washed with water by numerous decantations. The excess wash was then removed as far as practicable and the residue was carefully dried and assayed. The result showed a gold content of 1.8 dwt. per ton.

Three separate samples of the same material were then agitated with alkaline cyanide solution of graduated gold content. After settlement the supernatant solutions were filtered and assayed, and were found to have lost an amount of gold which was in strict proportion to their original gold content. These results confirmed the conclusions that the principal action under such circumstances is absorption, and not adsorption, and that the amount of gold-bearing cyanide solution abstracted by slime particles is by no means insignificant.

## CHAPTER XIX.

### THE DESIGN OF SPITZLUTTE AND SPITZKASTEN.

**General Principles.**—The action of a spitzlutte can be readily understood by those not already acquainted with it, by the diagrams, Figs. 29–31.

Fig. 29 shows a vertical tube with a column of water flowing through it in an upward direction. If a number of grains of various sizes and specific gravities are dropped through the funnel *a* into the ascending water, it is evident that the rate of flow may be so adjusted that some of the larger and heavier grains will fall downwards and settle at the bottom of the tube at *b*, while the lighter and smaller grains will be carried upwards and discharged with the water. And for each rate of flow there will be a corresponding set of

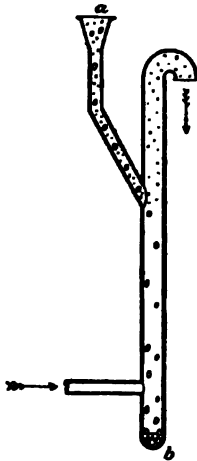


FIG. 29.—Action of the Spitzlutte.

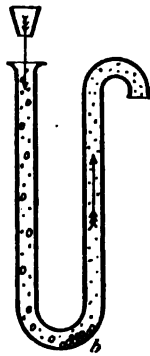


FIG. 30.—Action of the Spitzlutte.

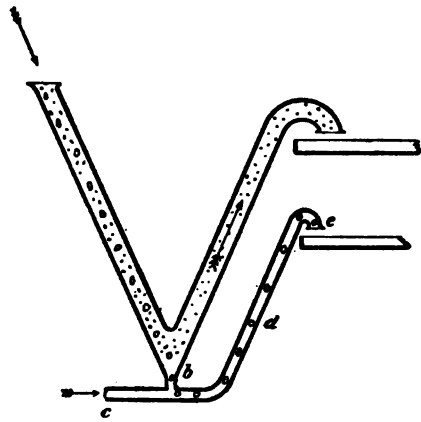


FIG. 31.—Spitzlutte.

grains which will just fall. Similarly, if battery pulp can be sent in an upward stream at a certain velocity through a pipe, some of the grains will be left behind and some carried forward by the ascending current. In a tube such as that shown in Fig. 30, for instance, the heavy, coarse sand will fall back and collect at *b* in the bend. Such an apparatus would, however, soon choke itself, but Fig. 31 shows a continuous arrangement in which there is a small outlet for coarse sand at the bottom of the bend. Naturally, some fine stuff and turbid water will flow out with the sand if no means be provided to prevent this. But by forcing in a small stream of clear water through

pipe *c*, the downward pressure of the slimy water is overbalanced and the coarse sand escapes alone in the clear water by pipe *d*. By regulating the size of the openings *b*, *c*, and *d*, and varying the height of the discharge *e*, a clean product can be obtained, the nature of which, however, depends essentially upon the velocity of flow in the main tube. Therefore, by repeating the operation with a series of decreasing velocities, several grades of sorted material can be obtained.

Mixed grains can also be sorted by means of a horizontally flowing stream of water, for, by sufficiently reducing its velocity, it will be unable to carry forward the largest and heaviest particles, which will consequently sink to the bottom. A further reduction of velocity will cause other finer grains to sink, until finally nothing but water flows away. This is the principle of the spitzkasten, which is merely a box, rectangular at the top, with sides and ends so inclined inwards that they come nearly to a point at the bottom. A small outlet is provided at the bottom for the escape of the falling grains. A series of such boxes, with successively increasing width, gives a set of roughly graded products.

**Theory of the Spitzlutte.**—The theory of bodies falling in water has been worked out by Rittinger, the inventor of the spitzkasten, and published in his *Lehrbuch der Aufbereitungskunde* from 1867 to 1873, and the reader is referred to that work for full details. Only a brief summary of his results can be given here, for the use of those to whom the original work is inaccessible.

Let  $W$  = pressure exerted on a flat horizontal surface by a vertically flowing stream of water whose velocity is  $v$ . Let  $f$  be the area of the surface.

**Then**

$$W = 51fv^2 \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

square metre of surface, a right-angled wedge will lose only one-half and an equilateral wedge one-quarter of this amount, on account of the upward motion of the water.

For a cone when  $\beta$  = half the angle at the apex,

$$a_2 = a \sin^2 \beta.$$

Therefore when

$$\beta = 45^\circ \quad a^2 = \frac{a}{2} \quad . \quad . \quad . \quad . \quad . \quad (4)$$

And for a hemispherical surface,

$$a_2 = \frac{a}{2} \quad . \quad . \quad . \quad . \quad . \quad (5)$$

So that in all cases, whether the body is a wedge, cone, or hemisphere, when the depth is half the breadth, the result is the same; that is to say,

$$a_1 = a_2 = a_3 = \frac{a}{2},$$

which for unit area and unit velocity = 25.5 kgm.

Rittinger, as the result of numerous experiments with spherical bodies from 1 to 10 mm. diameter, and of specific gravities from 1.3 to 11, gives the value of  $a_3$  at 25.5 kgm. very nearly; so that, within these limits, calculation and experiment agree very closely.

The above equations have been quoted, because they lead to the general conclusion that the resistance of the water to a falling body depends more upon the relative depth and width of it than upon the actual shape of its specific lower surface.

General formulæ for bodies falling in water are given as follows :—

$s$  = distance through which the body falls.

$t$  = time in seconds at the end of which the motion of the body is considered.

$d$  = diameter of sphere.

$\delta$  = specific gravity.

$\gamma$  = 1000 kgm. = weight of 1 cubic metre of water.

$$C = \sqrt{\frac{2\gamma}{3a_2}}.$$

Then

$$s = 5.11t \sqrt{d(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad (6)$$

And from this the mean velocity

$$v = \frac{s}{t} = 5.11 \sqrt{d(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad (7)$$

The following corollaries of the last equation are of importance:

(1) With two equal falling bodies of different diameters,  $d_1$  and  $d_2$  and different specific gravities  $\delta_1$  and  $\delta_2$ , then

$$\frac{d_1}{d_2} = \frac{\delta_2 - 1}{\delta_1 - 1} \quad . \quad . \quad . \quad . \quad . \quad (8)$$

That is to say, if the equal falling bodies have different densities, their diameters are also different, and *vice versa*, the denser body having the smaller diameter, and consequently the smaller volume. For quartz and ordinary iron pyrite, the diameters are as 2.5 to 1 nearly, and the volumes are as the cubes of these numbers, namely, about 16 to 1.

(2) If  $P_1$  and  $P_2$  represent the absolute weights of two equal falling bodies, they are thus related—

$$\frac{P_1}{P_2} = \frac{\delta_1(\delta_1 - 1)^3}{\delta_2(\delta_2 - 1)^3} \quad . \quad . \quad . \quad . \quad . \quad . \quad (9)$$

Thus, for quartz and pyrite the relative weights are about 10 to 1 as calculated from this last equation, and in all cases the denser body has not only a smaller diameter and volume, but also a smaller absolute weight.

These last considerations show clearly why it is possible to take out a large proportion of the pyrite from battery pulp, together with only the coarsest particles of the quartz, in a spitzlutte, as now so largely practised on the Rand, although, as will be seen later, the actual numerical results of the last two equations do not apply to very small grains of irregular shape, as shown by the experiments of R. H. Richards.

So far as already described, Rittinger's investigations deal only with bodies of regular shape. In treating of irregular bodies, he designates the resistance of the water by

$$\alpha_4 = \zeta \alpha = 51 \quad . \quad . \quad . \quad . \quad . \quad . \quad (10)$$

where  $\zeta$  is a coefficient, depending upon the form of the grains, which is determined experimentally.

Experiments with grains, selected according to shape as well as according to size, gave the following figures—

|                 |                |                  |
|-----------------|----------------|------------------|
| Rounded grains, | $\zeta = 1.25$ | $\alpha_4 = 65$  |
| Longish   ,,    | $\zeta = 1.86$ | $\alpha_4 = 90$  |
| Flat       ,,   | $\zeta = 2.35$ | $\alpha_4 = 120$ |

When the weight and sp. gr. of a grain are known, the diameter of a sphere of equal weight can be calculated. This 'ideal diameter' of the grain may be represented by  $d_1$ . Then the velocity of the falling grain is

$$v = \sqrt{\frac{2\gamma}{3\alpha_4} d_1 (\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (11)$$

Putting the constant factor

$$\sqrt{\frac{2\gamma}{3\alpha_4}} = C_2$$

then

$$v = C_2 \sqrt{d_1 (\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (12)$$

Assuming 50 per cent. of rounded grains and 25 per cent. of each of the other kinds, the average value of  $C_2 = 2.85$ , as compared with  $C_1 = 5.11$  for perfect spheres.

But Rittinger goes a step further in a practical direction, for instead of using this ideal diameter, he connects it with the size of sieve holes  $D$  by putting  $\mu D$  for  $d_1$  in the equation for  $v$ , thus—

$$v = \sqrt{\frac{2\gamma}{3\alpha_4} \mu D (\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (13)$$

Then putting the constant quantity

$$\sqrt{\frac{2\gamma\mu}{3\alpha_4}} = C_3,$$

he finally obtains the equation

$$v = C_s \sqrt{D(\delta - 1)} \quad (14)$$

and from experiments deduces the following values—

|                              |              |
|------------------------------|--------------|
| Rounded grains, $\mu = 0.73$ | $C_s = 2.73$ |
| Longish „ $\mu = 0.80$       | $C_s = 2.87$ |
| Flat „ $\mu = 0.67$          | $C_s = 1.92$ |

Average value, on the same assumption as before, as to the relative quantities of each shape,  $C_s = 2.34$ .

There is a slight discrepancy between the values given for  $\mu$  and  $C_s$  in the case of longish grains, but as it only affects the second place of decimals in the average value of  $C_s$  it is unimportant.

The average velocity for irregular grains having specific gravity  $\delta$ , and of the sieve class produced with holes of diameter  $D$ , can therefore be found from the following empirical equation—

$$v = 2.34 \sqrt{D(\delta - 1)} \quad (15)$$

Now, if the grains are placed in an upward current of water whose velocity is equal to the mean velocity of the falling grains, a state of equilibrium is produced for this particular kind of grain, and all grains which fall more slowly will be carried upward and thereby separated from those which fall more quickly, as already explained.

Rittinger sums up this part of his work in the following words:—

“The empirical values of  $C_s$  are nevertheless diverse, even with bodies of like forms; one must therefore despair of an exact formula for irregular bodies in water, and must be contented with the above approximation (*Schliefschlichen*), which is quite sufficient for practical work.”

He gives some tables calculated from the last equation, from which have been compiled the following figures. The diameters are given in inches, and the velocities in feet per second.

TABLE XXII.—Velocity required in upward flowing stream of water to hold irregular bodies in equilibrium.

| Sieve diameter,<br>in. | Velocity $v$ in feet per second. |                         |                          | Sieve diameter,<br>mm. = $D$ . | Velocity $v$ in metres per second. |                         |                          |
|------------------------|----------------------------------|-------------------------|--------------------------|--------------------------------|------------------------------------|-------------------------|--------------------------|
|                        | Galena<br>sp. gr. = 7.5.         | Pyrites<br>sp. gr. = 5. | Quartz<br>sp. gr. = 2.6. |                                | Galena<br>sp. gr. = 7.5.           | Pyrites<br>sp. gr. = 5. | Quartz<br>sp. gr. = 2.6. |
| 0.157                  | 1.286                            | 0.984                   | 0.640                    | 4                              | 0.392                              | 0.30                    | 0.195                    |
| 0.118                  | 1.115                            | 0.853                   | 0.558                    | 3                              | 0.340                              | 0.26                    | 0.170                    |
| 0.110                  | 1.073                            | ...                     | 0.528                    | 2.8                            | 0.327                              | ...                     | 0.161                    |
| 0.079                  | 0.912                            | 0.722                   | 0.449                    | 2                              | 0.278                              | 0.22                    | 0.137                    |
| 0.055                  | 0.761                            | ...                     | 0.377                    | 1.4                            | 0.232                              | ...                     | 0.116                    |
| 0.039                  | 0.640                            | 0.492                   | 0.318                    | 1                              | 0.195                              | 0.15                    | 0.097                    |
| 0.028                  | 0.541                            | ...                     | 0.269                    | 0.71                           | 0.165                              | ...                     | 0.082                    |
| 0.020                  | 0.453                            | 0.328                   | 0.239                    | 0.5                            | 0.138                              | 0.10                    | 0.073                    |
| 0.014                  | 0.380                            | ...                     | 0.190                    | 0.35                           | 0.116                              | ...                     | 0.058                    |
| 0.010                  | 0.321                            | ...                     | 0.157                    | 0.25                           | 0.098                              | ...                     | 0.048                    |
| 0.005                  | 0.226                            | ...                     | 0.112                    | 0.125                          | 0.069                              | ...                     | 0.034                    |



The next table shows the results obtained by R. H. Richards,<sup>1</sup> from an extensive series of experiments. As noted at the foot of this table, the diameters were obtained by measurement of sample grains, and not by the size of sieves as in Rittinger's experiments.

A comparison of these figures with those of Table XXII. shows a certain amount of disagreement, and to make clear the nature and extent of this,

TABLE XXIII.—*Diameters of Quartz and Galena particles which are equal settling in upward currents specified under free settling conditions.*

| Particles fall in<br>Current of inches<br>per second. | Particles rise in<br>Current of inches<br>per second. | Diameter of<br>Particles,<br>in. |         | Ratio between<br>diam. of Particles<br>actually obtained. | Diameter of<br>Particles,<br>mm. |         | Particles fall in<br>Current of milli-<br>metres per second. | Particles rise in<br>Current of milli-<br>metres per second. | Ratio between<br>diam. of Particles<br>averaged by curve. |
|---|---|----------------------------------|---------|---|----------------------------------|---------|--|--|---|
|   |   | Quartz.                          | Galena. |   | Quartz.                          | Galena. |  |  |   |
| ·000  | ·050  | ·00119*                          | ·00076* | 1·55  | ·0301*                           | ·0194*  | 0·00   | 1·26   | 1·54  |
| ·050  | ·099  | ·00132                           | ·00078  | 1·69  | ·0335                            | ·0198   | 1·26   | 2·51   | 1·68  |
| ·099  | ·199  | ·00224                           | ·00115  | 1·95  | ·0568                            | ·0292   | 2·51   | 5·05   | 1·82  |
| ·199  | ·292  | ·00304                           | ·00162  | 1·87  | ·0772                            | ·0412   | 5·05   | 7·42   | 1·96  |
| ·292  | ·394  | ·00387                           | ·00192  | 2·01  | ·0982                            | ·0488   | 7·42   | 10·01  | 2·09  |
| ·394  | ·577  | ·00561                           | ·00242  | 2·32  | ·1423                            | ·0613   | 10·01  | 14·68  | 2·23  |
| ·577  | ·780  | ·00739                           | ·00284  | 2·60  | ·1875                            | ·0721   | 14·68  | 19·80  | 2·35  |
| ·780  | 1·186   | ·0089                            | ·0041   | 2·18  | ·2254                            | ·1032   | 19·80  | 30·12  | 2·48  |
| 1·186   | 1·589   | ·0135                            | ·0051   | 2·62  | ·3416                            | ·1305   | 30·12  | 40·37  | 2·61  |
| 1·589   | 1·972   | ·0153                            | ·0055   | 2·76  | ·3880                            | ·1404   | 40·37  | 50·08  | 2·72  |
| 1·972   | 2·366   | ·0206                            | ·0067   | 3·07  | ·5241                            | ·1708   | 50·08  | 60·09  | 2·82  |
| 2·366   | 2·769   | ·0232                            | ·0079   | 2·95  | ·5892                            | ·1997   | 60·09  | 70·34  | 2·92  |
| 2·769   | 3·160   | ·0260                            | ·0094   | 2·77  | ·6590                            | ·2381   | 70·34  | 80·28  | 3·03  |
| 3·160   | 3·552   | ·0339                            | ·0108   | 3·13  | ·8604                            | ·2750   | 80·28  | 90·21  | 3·12  |
| 3·552   | 3·919   | ·0403                            | ·0135   | 2·99  | 1·0234                           | ·3428   | 90·21  | 99·54  | 3·21  |
| 3·919   | 4·384   | ·0450                            | ·0138   | 3·28  | 1·1424                           | ·3504   | 99·54  | 110·09   | 3·29  |
| 4·384   | 4·726   | ·0521                            | ·0144   | 3·62  | 1·3216                           | ·3648   | 110·09   | 120·03   | 3·36  |
| 4·726   | 5·135   | ·0560                            | ·0149   | 3·77  | 1·4224                           | ·3776   | 120·03   | 130·43   | 3·42  |
| 5·135   | 5·526   | ·0562                            | ·0166   | 3·39  | 1·4256                           | ·4208   | 130·43   | 140·37   | 3·49  |
| 5·526   | 5·918   | ·0632                            | ·0180   | 3·52  | 1·6032                           | ·4560   | 140·37   | 150·31   | 3·54  |
| 5·918   | 6·303   | ·0664                            | ·0181   | 3·67  | 1·6848                           | ·4592   | 150·31   | 160·09   | 3·59  |
| 6·303   | 6·691   | ·0689                            | ·0182   | 3·78  | 1·7488                           | ·4624   | 160·09   | 169·95   | 3·63  |
| 6·691   | 7·106   | ·0710                            | ·0207   | 3·44  | 1·8032                           | ·5248   | 169·95   | 180·51   | 3·66  |
| 7·106   | 7·826   | ·0778                            | ·0223   | 3·42  | 1·9744                           | ·5776   | 180·51   | 198·78   | 3·70  |

\* The length and width of each of ten grains of quartz and of galena were measured for each product by microscope micrometer, and the mean of each 20 measurements was taken as the average diameter.

in Table XXIV. is given, in parallel columns, a selection of figures from the two series. As Rittinger's velocities are for equilibrium, the mean of Richards' pair of velocities in each case fairly represents the same condition.

The first point to be noted is that for quartz grains of 1 mm. diameter, the two figures agree very closely, as there is less than 5 per cent. difference between them; and for galena grains of 0·35 mm. diameter, the difference is only 10½ per cent., reckoned on the smaller figure. It is probable, therefore, that the experiments from which Rittinger's coefficients were derived were

<sup>1</sup> "Spitzkasten and Slime Tables," *Proc. Am. Inst. of Min. Eng.*, vol. xxvii, p. 76.

carried out with grains of about these sizes. For both materials, however, Richards' results give a much greater range of velocities for the same range of diameters. In other words, Richards' figures for large grains show a greater velocity than Rittinger's, and for small grains a less velocity. It is clear, therefore, that there is some force in operation which increases in relative importance as the particles decrease in size, and which acts so as to retard the fall of the grains, and thereby aids the upward current, and it appears likely that this is due partly to the skin friction of the grains themselves. For this friction will tend to cause the grain to move with the water, because it opposes any relative motion between the two. Also the friction varies as the square of the diameter, while the weight varies as the cube thereof, so that the frictional resistance becomes relatively greater for small grains than for large ones. In Rittinger's investigation, skin friction was not taken into account, and it therefore requires modification to allow for this retardation of

TABLE XXIV.—*Comparison of Rittinger's and Richards' Results.*

| Diameter of Particle in millimetres. | Velocities for Quartz in metres per second. |           | Velocities for Galena in metres per second. |           |
|--------------------------------------|---|-----------|---|-----------|
|                                      | Rittinger.                                  | Richards. | Rittinger.                                  | Richards. |
| 2                                    | 0·137                                       | 0·192     | ...   | ...       |
| 1                                    | 0·097                                       | 0·093     | ...   | ...       |
| 0·5                                  | 0·073                                       | 0·053     | 0·138                                       | 0·167     |
| 0·35                                 | 0·058                                       | 0·036     | 0·116                                       | 0·105     |
| 0·25                                 | 0·048                                       | 0·028     | 0·098                                       | 0·077     |
| 0·125                                | 0·034                                       | 0·011     | 0·069                                       | 0·034     |

the fall of the grain. Pending further investigation in this direction, his equation is, however, useful for application in cases where experimental data are not available, and this comparative examination of it, in the light of Richards' results, serves to show in which direction it is likely to err in any particular case.

The column of ratios between diameters of quartz and galena grains in Table XXIII. shows that Equation 8 does not hold good numerically for small grains. Thus, the equation gives the relative diameters of equal-falling grains of quartz and galena as 4 to 1, whereas in the table the ratio varies from 3·78 to 1·55. In spite of certain irregularities which are unavoidable in such work, the gradual decrease in the ratio as the particles become smaller is very marked, and, on the whole, uniform. This point should also be borne in mind when using Equation 15 for determining the size of spitzlutte. This discrepancy between calculation and experiment is also probably due, to some extent, to the skin friction.

It would appear from the foregoing that gravity and skin friction were the only factors of importance that exert influence on bodies falling through water. This is the case with particles of an appreciable size; but as the

particles become finer, other factors, viz. viscosity of the water and electrostatic repulsion of the particles, become of even greater importance.

It is to be observed, however, that the principles elucidated by Rittinger's investigation are untouched by these discrepancies, and that a rising stream of water, as employed in a spitzlutte, is a suitable and fairly satisfactory means for (1) separating grains of equal density according to their size; (2) for separating equal-sized grains according to their specific gravity; and (3) for separating, from mixed bodies of different sizes and densities, a group of equal-falling grains.

The practice of speaking of pointed boxes as 'sizers,' is therefore inaccurate; they are distinctly hydraulic classifiers, capable of several kinds of classification.

Rittinger, in speaking of the products from spitzlutte and spitzkasten, says that "as the difference in volume will always be greater than the difference in weight, it may therefore be said that equal-falling bodies are similar in weight rather than similar in size." He uses the word 'class' to designate a group of grains of equal size, and the word 'sort' for a group of equal-falling bodies; but in English the latter word has a quite different and well-established meaning in metallurgy, so that it is not advisable to use it for this purpose.

**Practical Design of Spitzlutte.**—For a good separation, it is necessary that the rising stream should have, as nearly as possible, the same velocity at every part of its cross-section. This desideratum is attained by passing it through a rectangular passage, one of the dimensions of which is small relatively to the other. Thus, in Fig. 32 the passages *a* and *b* are 18 in. by 1½ in. The pulp arriving by the launder *l* passes down passage *a* and then upwards through *b* to the overflow at *m*. The selected grains therefore fall out of the flowing stream at *c*, and below this point the space *n*, with inclined sides *d d*, is provided to lead the separated product to the central discharge pipe *o*. Clean water is supplied through pipe *c* at sufficient pressure to keep back the battery water and the finer solids, and so to ensure a clean product. In a series of apparatus of this kind, the second one would have a passage of larger sectional area, in order to obtain less velocity, and consequently a finer product.

From the data in the above tables it is now easy to find the proper dimensions of the passage for a given quantity of pulp, in order to separate grains of a certain size of any substance whose specific gravity is known.

As an example, assume that a spitzlutte is required to separate, from a battery pulp, the coarsest particles of quartz which have passed through a 25-mesh screen, with holes of, say, 0.026 in., and that 250 tons of ore are crushed daily. At 9 tons of water per ton of ore, the total quantity of pulp is 74,575 cu. ft. per day, or 0.863 cu. ft. per sec. From Table XXIII, the necessary velocity is found to be 0.231 ft. per sec. The required sectional area is equal to

$$\frac{\text{quantity}}{\text{velocity}} = \frac{0.863}{0.231} = 3.74 \text{ sq. ft.},$$

or 538 sq. in. By taking, say, 4 in. as the small dimension of the passage, its width must be 11 ft. 2½ in. This width would require a great depth for the triangular space ( $n$  in Fig. 32) below the passage, and it is therefore more convenient to divide it between, say, three boxes, each of which will then be 3 ft. 9 in. wide. Of course, with the quartz grains of the selected size, the product will also contain smaller particles of all the heavier minerals present, and, therefore, a much greater percentage of mineral than the original pulp.

The next step is to determine the difference between the levels of inflow and outflow, which is required to cause the pulp to pass through the

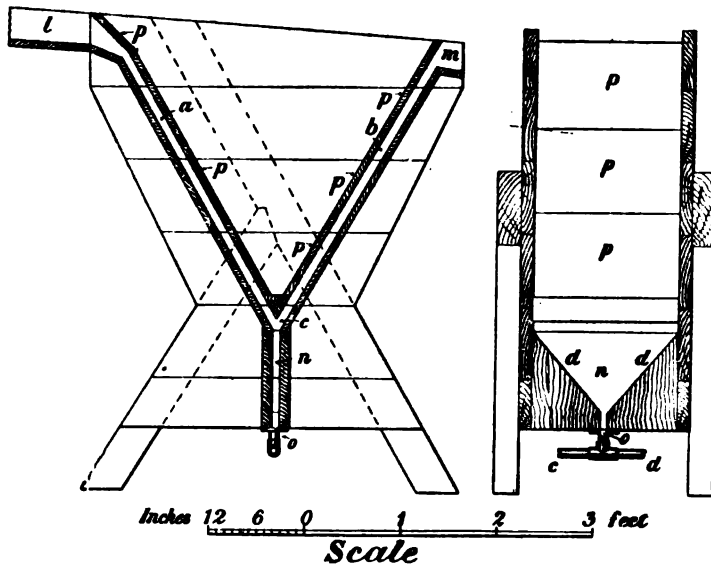


FIG. 32.—Spitzlutte.

passage at the velocity for which the spitzlutte is designed. The ordinary formula  $h = \frac{v^2}{2g}$  may be used, and extra head allowed for friction in the passage, but in practically all cases the calculated head is so small, say ⅛ in. or thereabout, that it is negligible in comparison with that required to overcome the friction, which may be from ½ to 2 in., according to the size of the apparatus and the material of which it is built.

As already mentioned, the object of making one dimension of the passage small is to secure a uniform product; but the above example shows that this requires sometimes a great width when the apparatus is constructed with a single passage, as in Fig. 32.

Fig. 33 shows diagrammatically how this objection may be obviated by the use of several parallel passages,  $a$  and  $b$ .

But it often happens that extreme accuracy of grain selection is not required, and in such cases the cross-section of the spitzlutte passage may

approximate to a square. Thus, in the above example, by making the smaller dimension 18 in. the width is reduced to 2 ft. 6 in. for the whole of the pulp, and the depth is also thereby sensibly diminished. In this case the apparatus has the general form shown in Fig. 34.

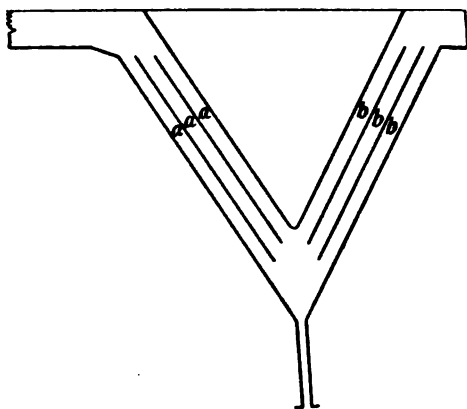


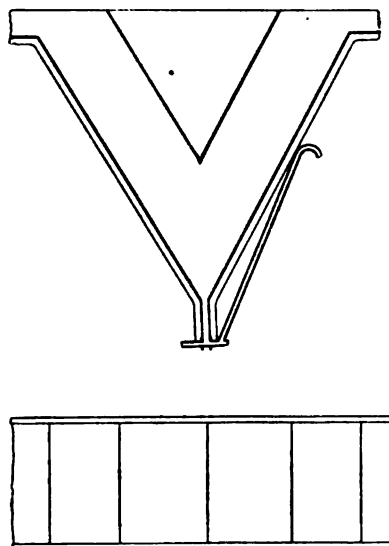
FIG. 33.—Spitzlutte.

Fig. 35 illustrates a neat form of spitzlutte. The cross-section of the passage can be adjusted to give any required product within limits, by raising or lowering the inner V by means of the screw and nut shown. The pressure water is admitted through a narrow slit which extends across the width of the apparatus.

Another stage in the modification of the spitzlutte is shown in Fig. 36. In this form the inner V is replaced by a vertical partition *e*, arranged to slide up or down to effect a rough regulation of the product. This form involves a serious departure from theoretical perfection of product, because the velocity of the stream will not only be different at various portions of any cross-section, but will also vary at different parts of the length of the passage, being slower at *a* and *b* than it is at *c* and *d*. In fact, this last form is really intermediate between the spitzlutte and the spitzkasten, for as the partition *e* is raised, the definite control of the flow is gradually diminished, until, with the entire removal of the partition, the transition of the apparatus to an actual spitzkasten is complete.

**Spitzkasten.**—This form of apparatus does not lend itself so readily to calculation as does the spitzlutte; and the rules for determining its shape and size are chiefly founded upon the results of experience. Rittinger recommends the use of four boxes in succession, the width of each being twice that of the preceding one. For the first, his rule is to allow a width of one-tenth of a foot for each cubic foot of pulp flowing per min., i.e., 6 ft. for each cubic foot per second.

Taking, for example, as before, a battery crushing 250 tons per day, i.e.,



Scale of Feet.  
0 1 2 3 4 5 6

FIG. 34.—Spitzlutte.

0·863 cu. ft. per sec., the widths of the four boxes are 5 ft. 2 in., 10 ft. 4 in., 20 ft. 8 in., and 41 ft. 4 in.

Linkenbach, in 1887,<sup>1</sup> gave the following rule: The width of each box is to be  $1\frac{1}{2}$  times that of the preceding one, and for the first box an allowance of 0·067 is to be made for each cubic foot per min., i.e., 4 ft. for each cubic

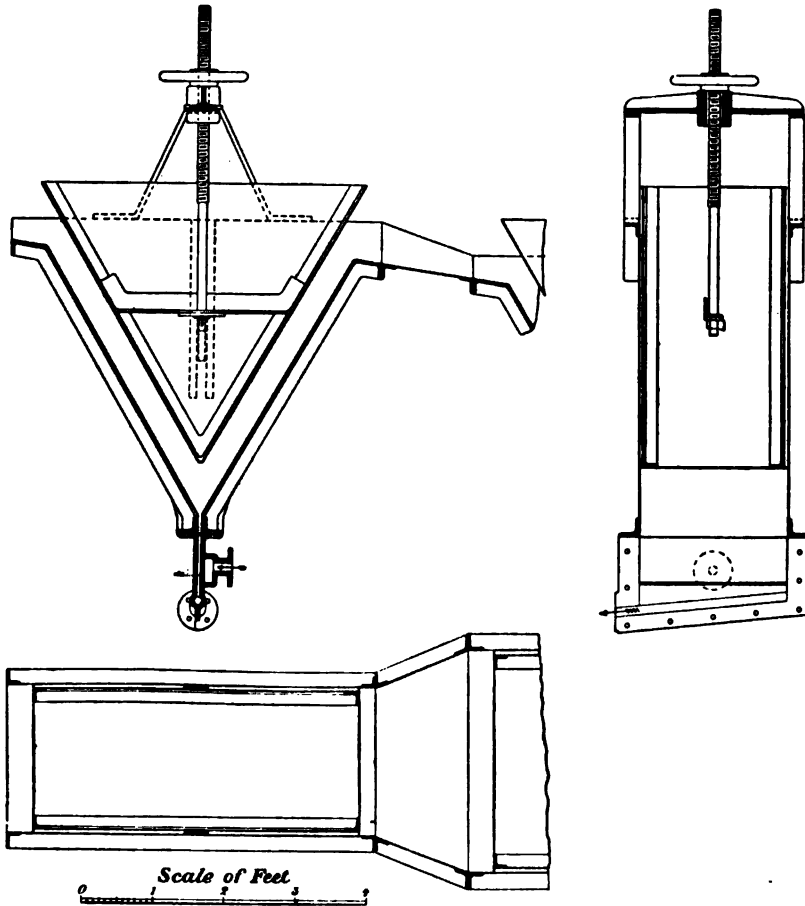


FIG. 35.—Spitzlutte.

foot of pulp per sec. These sizes are said to be suitable for material from a  $1\frac{1}{2}$ -mm. screen (0·059-in.), when three boxes are used in series for sand. He further recommends the use of three boxes for slime, of which the first is to have a width of 0·213 ft. for each cubic foot per min., or 12·78 ft. for each cubic foot per sec. The width of the last box will, by this rule, be 24 ft. 10 in. There is a considerable difference in these two sets of figures, due probably to the different conditions as regards the material treated and the

<sup>1</sup> *Die Aufbereitung der Erze*, Berlin, 1887.

water used. The presence of acid in the water, or of carbonate of lime in the ore, would facilitate the settlement of the finer particles.

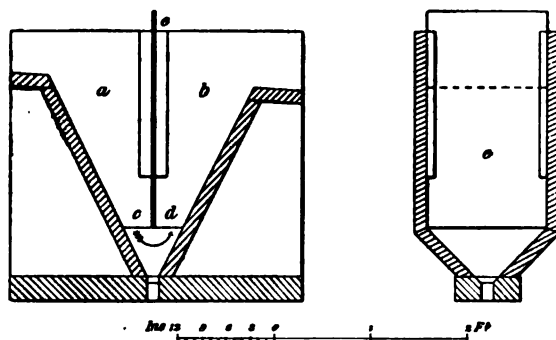


FIG. 36.—Spitzlutta.

In compiling the following table, the figures of both writers have been compared, as to the sizes of the grains for which the boxes were intended, with the practice on cyanide plants, so far as reliable figures are available.

TABLE XXV.—*Width of Spitzkasten for separating Grains of Quartz.*

| Size of Grain required in the Product.               | Width for each cubic foot of Pulp per minute.<br>ft. |
|--|--|
| Coarse grit, . . . . .                               | 0·05   |
| Sand passing sieve with holes = 0·05 inch, . . . . . | 0·10   |
| „ 30-mesh sieve, . . . . .                           | 0·20   |
| „ 40- to 60-mesh, . . . . .                          | 0·30   |
| „ 90- to 120- „, . . . . .                           | 0·40   |
| Slime from Rand basket with lime, . . . . .          | 0·55   |
| Slime settlement without lime, from . . . . .        | 0·60   |
| „ „ „ to . . . . .                                   | 2·50   |

The length of the box is of less importance. Rittinger recommends 6, 9, 12, and 15 ft. for his four boxes, while Linkenbach, for six boxes, gives the following sizes: 1 ft. 8 in., 2 ft. 6 in., 3 ft. 9 in., 12 ft., 16 ft., and 20 ft.

**Inclination of Walls.**—If the walls are not sufficiently steep, the falling grains collect in heaps, especially in the angles of the inverted pyramids, and fall occasionally in masses upon the outlets, which thereby become blocked. Experience shows that an angle of 55° with the horizontal is desirable, and only in extreme cases of limited fall should the absolute minimum of 45° be allowed, when the walls should be lined with glass or other smooth material. The depth inside will therefore, in ordinary cases, be equal to  $\frac{B}{2} \tan 50^\circ = 0·6B$ , where B is the greater longitudinal inside dimension.

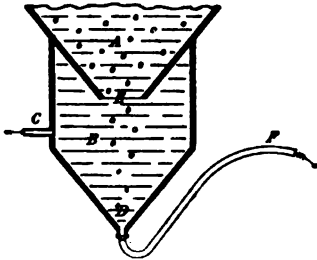


FIG. 37.—Discharge orifice.

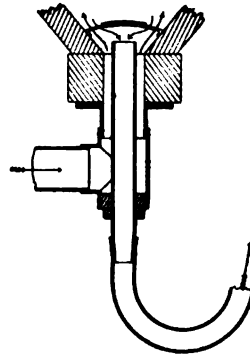


FIG. 38.—Discharge orifice.

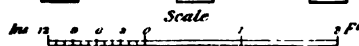
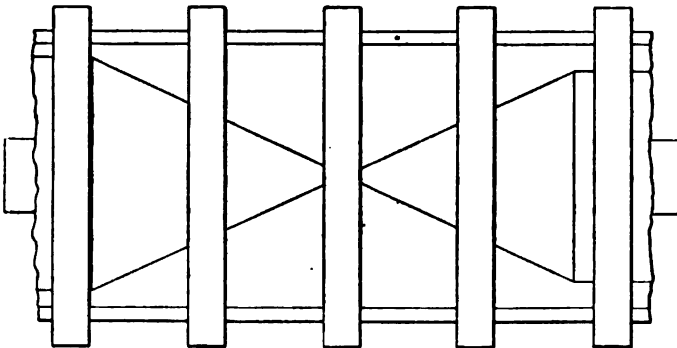
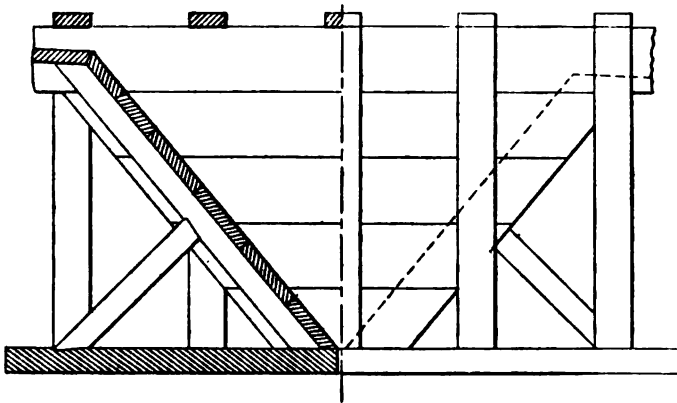


FIG. 39.—Spitzkasten.



**Discharge Orifice and Pressure Water.**—The discharge, when no pressure water is used, is proportional to the square root of the depth, and therefore to  $\sqrt{0.6B}$ . But the quantity of pulp passed through the box, and therefore also the amount of solid matter discharged of the selected size, is proportional to the width. Consequently, with an orifice of given size, a large box should give a thicker product than a small one. The minimum size for the discharge necessary to obviate choking is about  $\frac{3}{4}$  in. diameter.

The sketch given in Fig. 37, which is a form sometimes used, illustrates the functions of the pressure water.

A is the lower part of the spitzkasten, B is a separate chamber, C the pipe supplying pressure water, and D the outlet for the separated product. An ideal case may be assumed where the pressure in B exactly balances the downward pressure at the opening E. Then the selected grains would (theoretically) fall from the muddy water in A through E into the clear water in B, and so through the outlet D, by which only clean water and selected grains would pass. To ensure this last condition in practice, the pressure in B has to be more, say 6 to 12 in., than that in A, and to avoid an excessive waste of clean water a counter-pressure has to be set up at D. This is done by attaching a bent pipe F, the outlet of which is at a higher level.

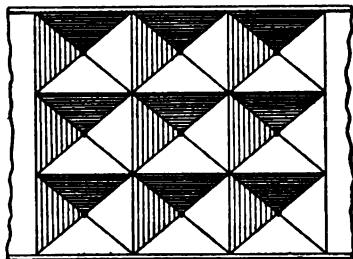


FIG. 40.—Spitzkasten in plan.

By using a flexible hose for this purpose, the outflow can be immediately regulated at any time by raising or lowering the upper end of it. For coarse material, the net difference between the overflow level and the underflow discharge may be from 3 to  $3\frac{1}{2}$  ft., and for slime from 2 to  $2\frac{1}{2}$  ft.

Fig. 38 shows a discharge pipe arrangement that has given good results in practice, and that shown in Fig. 39 is also suitable.

With care, the outflow may be made to contain as much as 37 per cent. of solid matter from the first boxes, but for slime, as little as 7 per cent. is considered satisfactory in some cases.

The spitzkasten is of simple construction, and will be readily understood from the example given in Fig. 39.

As it is often impossible to use large spitzkasten consisting of a single pyramid, on account of their great depth and the consequent fall required at the site, the arrangement of a number of small pyramidal boxes, placed side by side and end to end, as indicated in Fig. 40, is commonly adopted where large quantities of pulp have to be classified.

## CHAPTER XX.

### THEORY OF EXTRACTION BY SUCCESSIVE WASHINGS.

In this section it is proposed to demonstrate a few general rules which apply not only to decantation treatment, but to filter-press washing, and also to intermittent leaching in vats when they are drained to an equal degree after the application of each successive solution.

As the actual weight of dry slime treated is the most convenient standard with which the quantities of solution can be compared, this will be taken as equal to unity.

Then let  $s$  = total amount of liquor dealt with, including any water present in the slime before treatment.

Let  $m$  = amount of liquor retained by the slime after each decantation.

It will be further assumed that all the available gold is dissolved during the first agitation, the sole object of the subsequent operations being merely to separate as much as possible of this from the slime. It is evident, in the first place, that the fraction of this total dissolved gold which is extracted at each operation is equal to the fraction of the liquor which is drawn off.

Thus, when the whole solution is assumed to be all applied at one time, so that there will be only one decantation, the fraction of gold extracted will be

$$\frac{s-m}{s} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

It may be noted here that if the slime is quite dry when first placed in the vat, the total solution added is equal to  $s$ , but if the slime has been settled by water in the vat, and the clear water drawn off before the treatment begins, it will then contain  $m$  volumes of water, and consequently the solution to be *added* to wet slime will be  $s - m$  volumes only. In either case the above expression is true, it is assumed, for the sake of simplicity, that each solution or wash used contains no gold when applied to the slime.

In the case where only part of the total solution is applied at first, and the remainder in one or more successive portions:

Let  $a, b, c, \dots, l$  denote the respective quantities of liquor drawn off at each operation; then the quantity of liquor present in the vat *before* each decantation will be  $a+m, b+m, c+m, \dots, l+m$  respectively.

At the first decantation, the fraction of liquor drawn off, and consequently the fraction of gold extracted, is

$$\frac{a}{a+m} \cdot \dots \cdot \dots \quad (2)$$

and the gold remaining in the vat is

$$1 - \frac{a}{a+m} = \frac{m}{a+m} \quad (3)$$

At the second decantation of the gold *then present* the fraction

$$\frac{b}{b+m}$$

will be extracted ; therefore the fraction of total gold which is obtained by this second decantation is

$$\frac{m}{a+m} \times \frac{b}{b+m} = \frac{bm}{(a+m)(b+m)} \quad (4)$$

and the fraction of total gold which remains in the vat will be

$$\frac{m}{a+m} - \frac{bm}{(a+m)(b+m)} = \frac{m^2}{(a+m)(b+m)} \quad (5)$$

Proceeding in the same way, the fraction extracted at the third decantation is

$$\frac{cm^2}{(a+m)(b+m)(c+m)} \quad (6)$$

and the fraction remaining in the slime is

$$\frac{m^3}{(a+m)(b+m)(c+m)} \quad (7)$$

Finally, if  $l$  is the quantity of liquor drawn off at the last operation, and there are  $n$  decantations altogether, then :

The fraction extracted by the  $n$ th decantation is

$$\frac{lm^{n-1}}{(a+m)(b+m)(c+m) \dots (l+m)} \quad (8)$$

and fraction of total dissolved gold left in the residues is

$$\frac{m^n}{(a+m)(b+m)(c+m) \dots (l+m)} \quad (9)$$

From the above expressions the total extraction can be calculated in two ways, either by adding together the separate extractions, namely (2), (4), (6), . . . . (8), or by subtracting the residual gold in (9) from unity. By adopting the latter method, the total extraction after the  $n$ th decantation, which may be called  $E$ , then :

$$E = \frac{(a+m)(b+m)(c+m) \dots (l+m) - m^n}{(a+m)(b+m)(c+m) \dots (l+m)} \quad (10)$$

Now, it is evident that as  $a, b, c$ , etc., are positive quantities, the expression  $(a+m)(b+m)(c+m) \dots (l+m)$  is always greater than  $m^n$ , and that therefore the maximum value of  $E$  is coincident with the maximum value of that expression. But the sum of these factors is equal to  $s - m + mn$ , which in any given case is a constant quantity. Consequently,  $(a+m)(b+m)(c+m) \dots (l+m)$  has its maximum value when all its factors are equal, and

one useful, practical conclusion results, viz.—*With any quantity of solution and any number of decantations, the maximum extraction is obtained when equal quantities of solution are decanted at each operation.*

With equal decantations let  $q$  = the quantity of liquor drawn off at each operation, then

$$q = \frac{s-m}{n},$$

and therefore in this case equation (10) becomes

$$E = \frac{(q+m)^n - m^n}{(q+m)^n} = \frac{\left(\frac{s-m}{n} + m\right)^n - m^n}{\left(\frac{s-m}{n} + m\right)^n} \quad (11)$$

From this the total theoretical extraction can be calculated for any given values of  $q$ ,  $m$ , and  $n$ , or  $s$ ,  $m$ , and  $n$ .

And again, because  $\frac{s-m}{n} + m$  is necessarily greater than  $m$ , it follows that with a fixed value of  $s$  and increasing values of  $n$ , the value of  $E$  will continually increase, always approaching, but never reaching, unity. Therefore *a given total quantity of liquor will give the highest extraction when divided into the largest practicable number of separate washes.*

By careful settlement and decantation in the laboratory, using sufficient lime, the moisture remaining can be reduced to about 30 per cent. of the pulp remaining in the vat, which gives the value,  $m = 0.43$ ; but in practice it is generally found that, after decantation, the remaining mass contains 50 per cent. of water. In this case  $m = 1$ , and from equation (11):

$$E = \frac{(q+1)^n - 1}{(q+1)^n} = \frac{\left(\frac{s+(n-1)}{n}\right)^n - 1}{\left(\frac{s+(n-1)}{n}\right)^n} \quad (12)$$

As it is usual to express results in percentages, let  $P$  represent the total extraction per cent. of the dissolved gold, then—

$$P = 100E \quad (13)$$

From the above equations, Tables XXVI. and XXVII. have been calculated for the values  $m = 1$  and  $m = 0.75$ , respectively, in order to exhibit in a clear and concise manner the effect of varying the number and quantity of the solutions, under all the conditions which are likely to occur when working with natural settlement.

The first fact exhibited by Table XXVI. is that, although with any given quantity of liquor, as already stated, the extraction can be always increased with increased subdivision of the liquor, yet when small quantities are used, the rate of increase is so slow that a good extraction cannot be obtained with any reasonable number of decantations. For instance, with 2 tons of liquor per ton of dry slime, divided into 5 parts, the total extraction after the fifth decantation is only 60 per cent., and to reach a theoretical extraction of

95 per cent. it would be necessary to divide the solution into 200 parts, with 200 decantations, which of course is impracticable.

TABLE XXVI.—*Showing the Total Percentage of Dissolved Gold which is extracted from Slime by n equal decantations, when the residual pulp contains 50 per cent. of moisture, i.e. when  $m = 1$ .*

$s$  = total quantity of liquor used, including original moisture.

| Value of $s$ . | Number of Decantations. |       |       |       |       |
|----------------|-------------------------|-------|-------|-------|-------|
|                | 1                       | 2     | 3     | 4     | 5     |
| 2              | 50                      | 55.56 | 57.81 | 59.02 | 60    |
| 3              | 66.67                   | 75    | 78.4  | 80.25 | 81.41 |
| 4              | 75                      | 84    | 87.5  | 89.34 | 90.48 |
| 5              | 80                      | 88.89 | 92.18 | 93.75 | 94.71 |
| 6              | 83.33                   | 91.84 | 94.73 | 96.10 | 96.88 |
| 7              | 85.71                   | 93.75 | 96.30 | 97.44 | 98.06 |
| 8              | 87.5                    | 95.06 | 97.30 | 98.25 | 98.75 |
| 9              | 88.89                   | 96    | 97.97 | 98.77 | 99.16 |
| 10             | 90                      | 96.69 | 98.44 | 99.10 | 99.42 |
| 11             | 90.91                   | 97.22 | 98.77 | 99.33 | ...   |
| 12             | 91.67                   | 97.63 | 99.02 | ...   | ...   |

Here, then, is another reason why comparatively large volumes of solution must be used when treating slime by natural settlement. The table shows, however, that with 6 tons of liquor per ton of slime and four decantations, or with 10 tons divided into two parts, an extraction of over 96 per

TABLE XXVII.—*Showing Percentage Extraction when  $m = 0.75$ , or about 43 per cent. of moisture in residue after each decantation.*

| Value of $s$ . | Number of Decantations. |       |       |       |       |
|----------------|-------------------------|-------|-------|-------|-------|
|                | 1                       | 2     | 3     | 4     | 5     |
| 5              | 85                      | 93.19 | 95.85 | 97.07 | 97.74 |
| 6              | 87.5                    | 95.06 | 97.3  | 98.25 | 98.74 |
| 7              | 89.29                   | 96.25 | 98.14 | 98.89 | 99.26 |
| 8              | 90.63                   | 97.06 | 98.67 | 99.27 | 99.76 |
| 9              | 91.67                   | 97.63 | 98.97 | 99.49 | ...   |

cent. can be obtained. And these figures may safely be said to represent the extreme practical limits of the method of decantation after natural settlement.

**Reduction of Residual Moisture.**—A comparison between Tables XXVI. and XXVII. shows the advantage gained by a reduction in the amount of water retained by the slimes after settlement and decantation; in the latter table this quantity is equal to 0.75 of the dry weight of the slime. That is

to say, the residual pulp only contains about 43 per cent. of moisture after each operation, instead of 50 per cent. as in Table XXVI. Referring again to the three sets of conditions (*a*, *b*, and *c*) selected above, the increases in the extraction are 2.15, 1.84, and 1.63 per cent. respectively. The gain, therefore, is not great, and is relatively smaller with the larger quantities of liquor, but it is sufficient to encourage efforts to reduce the quantity of residual moisture, especially in the case of rich slime.

Although the preceding results are useful for the purpose of comparing the respective advantages of different methods of working, yet the fact must not be overlooked, that they are only strictly accurate when the liquor used for washing contains no residual gold from previous operations. But in practice, of course, the solutions generally contain some gold, and it is necessary to examine the loss thereby incurred, on the assumption that all the liquor used in dissolving and washing has the same gold content when applied to the slimes.

If the slime is dry when put into the vats, this loss is independent of the number of decantations and of the total quantity of liquor, and depends only upon the quantity of solution *m* which is finally left in the slime when the treatment is completed. Thus if *m* = 1, then for every grain of gold per ton in the wash liquor, one grain will be lost per ton of slime treated.

When treating wet slime, the loss is always less, because of the original moisture, a part of which remains to dilute the final wash. The larger the number of decantations, the more nearly will the loss approach to its greatest limit, which is identical with the amount lost in treating dry slime.

The following equations will enable the amount of this loss to be readily calculated for any values of *m* and *q*, and for one, two, or three successive operations, which will include all practical cases. The following equations relate *only* to the gold in the solutions before use :

Let *x* = assay of solution added to the slime.

*q* = as before, the quantity decanted each time.

*m* = as before, the moisture remaining each time.

Then quantity of gold added to slime with each wash liquor = *qx* ; and, of this, *x* is returned at first decantation,

$$\frac{qx}{1} \times \frac{q}{q+m} = \frac{q^2x}{q+m} \quad . \quad . \quad . \quad . \quad . \quad . \quad (14)$$

And left in slime after first decantation,

$$qx - \frac{q^2x}{q+m} = \frac{mqx}{q+m} \quad . \quad . \quad . \quad . \quad . \quad . \quad (15)$$

Returned at second decantation,

$$\frac{q^2x(q+2m)}{(q+m)^2} \quad . \quad . \quad . \quad . \quad . \quad . \quad (16)$$

Left in slime after second decantation,

$$\frac{mqx(q+2m)}{(q+m)^2} \quad . \quad . \quad . \quad . \quad . \quad . \quad (17)$$

Returned at third decantation,

$$\frac{q^2x(q^2 + 3mq + 3m^2)}{(q + m)^2} \quad . \quad . \quad . \quad . \quad . \quad . \quad (18)$$

Left in slime after third decantation,

$$\frac{mqx(q^2 + 3mq + 3m^2)}{(q + m)^2} \quad . \quad . \quad . \quad . \quad . \quad . \quad (19)$$

One example may be given:—Assume that 3 tons of liquor, assaying  $1\frac{1}{2}$  grains, is added and decanted twice, then the loss of gold from this source per ton of slime when  $m = 1$  is by (17)—

$$\frac{1 \times 3 \times 1.5(3 + 2)}{4 \times 4} = 1.4 \text{ grains.}$$

## CHAPTER XXI.

### THE COMMINUTION OF GOLD- AND SILVER-BEARING ORES.

It does not come within the scope of a textbook on cyanidation to deal with the mechanical considerations involved in the crushing, milling, or fine grinding of ores. The success or failure of the cyanide process depends, almost entirely, on the ultimate recovery of the metals, and this in turn is affected by the efficiency of the preliminary grinding or pulverising operations which are carried out in order that the ore may be in a suitable condition for cyaniding. The question, however, is not alone one of recovery, because it is necessary to take into consideration the fact that comminution involves one of the greatest expenses the metallurgist has to consider in the treatment of an ore. If comminution is carried beyond an economic limit, then increased cost counterbalances increased recovery. No definite rule can be laid down, and generalisations should be avoided. Medium crushing, followed by the leaching of the greater proportion of the tonnage, is still the predominant practice in the case of the vast bulk of simple gold ores. By this method it is possible to handle large tonnages at minimum cost, thus permitting the application of the process in the case of very low-grade ores.

Mistaken ideas sometimes lead to the adoption of excessively fine grinding, irrespective of the question of economic result, in order to be strictly in line with what is usually termed up-to-date practice. All-sliming, however, is only up to date in that it was found absolutely imperative in the most recent adoption of cyaniding, *i.e.* in the case of silver ores. It was practised in Western Australia as far back as 1892 in the treatment of gold ores. In both these instances all-sliming was adopted on the grounds of expediency as well as necessity. It remains to remark that no good will come from the adoption of any particular scheme of comminution for no other reason than successful application elsewhere. Each ore has its own peculiar characteristics and, away from the Rand, there is no justification for the mere duplication of existing equipment or the imitation of methods which, although suitable in one case, may be altogether unsuitable in another.

In the comminution of a gold or silver ore the object aimed at is to break the ore into such a size that the surface of the gold and silver is brought into contact with the cyanide solution. This end is attained by pulverisation, and much may be learnt by examining the grains under a low-power microscope to determine the minimum degree of fineness necessary, a point which, however, must be ultimately decided from empirical results



by testing with cyanide solution. The more finely the ore is crushed the more perfectly does the gold and silver dissolve, but if crushed too fine an excess of dust or slime forms. With an increase in the percentage of slime or slimed ore there is a decrease in the recovery, by displacement, of the gold and silver in solution, and the consumption of cyanide is generally higher. If crushed too coarse, a large percentage of the gold and silver remains locked up and is not dissolved, but that portion which does dissolve may be separated, more or less readily. Under no circumstances is it possible to crush an ore to one uniform size, but the aim should be to crush so as to get a maximum number of those particles that give the highest extraction at the lowest cost. This can be found only by experiment.

The size to which an ore has been crushed for the best results naturally varies with the character of the ore, and with the minerals which are intimately associated with the precious metals. Thus at the Mercur mine, Utah, where the ore is of a porous nature, particles of half inch or larger have been treated for the best results, whereas in Western Australia the sulpho-tellurides are crushed to a fine powder in order that the gold may be dissolved by the cyanide. Sometimes oxidised ore mined near the surface disintegrates or cleaves readily along lines containing the precious metals, in which case only coarse crushing is necessary to give the most economical results. In the case of simple gold ores it is, however, desirable in most cases to crush so that by far the greater proportion of the product passes a 60-mesh screen, with the production of a minimum of slime or dust. With silver ores, 'all-sliming' is practically imperative as it is, in some cases, with refractory gold ores.

Then, again, in many cases it is advantageous to crush wet, while in other cases dry crushing is found more beneficial. Local conditions and the nature of the ore have chiefly to do with the choice of methods, and it is only by systematic experiments on a commercial scale that the most economical means and the best degree of fineness can be determined.

**Dry and Wet Crushing.**—The time has long since passed when discussions on the merits or demerits of either method were common; and it is now generally realised that the class and grade of the ore alone affects decision. When a gold ore, on account of refractory contents, has to be roasted before cyanide treatment can be effectual it is obvious that dry crushing is economically imperative in the first instance. In exceptional cases an ore may be of such a grade and the gold in such a condition that comparatively coarse crushing, followed by direct leaching treatment of the whole product, produces the best net results. In this case, again, dry crushing is favoured on account of the fact that the ground ore is not classified in any way by gravitational settlement before it reaches the mixer or leaching vats, and percolation of the entire mass is feasible.

On the other hand, an ore may be treated entirely by cyanide and may contain metal in such quantity that it is imperative that the solvent solution be brought in contact with the ore at the earliest possible moment. This is

the case with most silver ores, and forms the main reason why wet crushing is invariably adopted.

Again, an ore may contain a considerable amount of gold in an amalgamable form. In this case wet crushing with lime water is advisable in order to coagulate the slime, cleanse the gold, and obviate or reduce the expense of mixing and conveying.

Amalgamation and cyaniding are complementary processes in the treatment of a gold ore; and there is no question of antagonism between the degrees of comminution required for the one or the other. Indeed it has been proved of late that excessive comminution of the ore may result in a decreased net profit as well as a reduction in the amalgamation recovery percentage. In any case the advantages of amalgamation are almost always available, as the process may be applied at any stage in the comminution of the ore. The maximum extraction by cyanide is invariably reached with a finer product than that required for efficient or maximum amalgamation; and even if all-sliming is necessary it may be carried out after the amalgamation of the ore in a coarser condition.

## CHAPTER XXII.

### MILLING IN CYANIDE SOLUTION.

IN the year 1893 the practice of milling in cyanide solution was introduced by the African Gold Recovery Company, at the May battery, near Johannesburg. The mortar boxes were supplied with solution; and the pulp, after amalgamation on the plates, was conveyed directly to the leaching vats. The solution, when partly cleared of slime, was returned to the battery. The process was abandoned after a short trial, but the cause of failure was not officially stated. Milling in cyanide solution has, however, since been adopted in a number of instances. The discussion of the subject which follows has been abstracted from a monograph on the subject.<sup>1</sup>

**Advantages of Milling in Cyanide.**—One of the chief advantages accruing from the practice of milling in cyanide solution is the intimate contact resulting between coarse metal and solvent, caused by the action of the grinding machinery. This advantage shrinks in importance in the case of amalgamable ore when consideration is paid to the fact that milling in cyanide precludes the possibility of amalgamation being practised, or renders its functions only partly effective. If any quantity of the metal in an ore can be obtained by a simple and direct method, an explanation is needed for the substitution of a complicated alternative which actually recovers only a proportion.

**Amalgamation and Cyaniding.**—In the case of gold ores, and in spite of clear enunciations of the actual facts disclosed by milling results, a good deal of misapprehension exists; and the question has been complicated and the issue beclouded by reference to the efficiency of amalgamation with unamalgamable gold ores. Such considerations do not affect the question. If a proportion of the gold in an ore is amalgamable, then practically 100 per cent. of this proportion is recoverable. If cyanidation replaces amalgamation as a treatment process for this proportion, then only from 90 to 95 per cent. is usually recovered.

The arguments advanced in favour of part amalgamation have only been used as referring to amalgamable gold; and it is therefore an untenable claim to advance that this gold may not amalgamate. In special but exceptional cases the recovery of gold by cyanidation may reach 97 per cent., but in no single instance has a recovery of 100 per cent. been recorded with any professional claim to verisimilitude. These facts may be substantiated by milling barren material, similar in physical characteristics to the ore usually

<sup>1</sup> "Milling in Cyanide Solution," by A. W. Allen, *Met. and Chem. Eng.*, May 15, 1918.

found in gold treatment plants, and mixing with this a proportion of coarse gold recovered by the amalgamation process. No matter how fine the ultimate division of the mixture, there will remain a small percentage of the original amount of gold associated with the gangue after complete milling-in-cyanide and filtration treatment.

**Extraction versus Recovery.**—A second method of beclouding the issue is to draw attention to the total amount of gold dissolved or extracted by cyanide and to ignore the question of recovery. Finely divided gold may be completely dissolved in a cyanide solution and thus completely extracted from the ore: but when associated with gangue as an average-grade material, such extracted gold cannot be completely recovered by any known combination of metallurgical processes operating on a working scale. All or nearly all the gold may be dissolved and a high percentage may be recovered as bullion. The loss of the remainder is all that is needed to influence the contention in favour of efficient amalgamation of gold ores previous to cyanide treatment.

**Colloids and Absorption of Metal.**—Absorption of dissolved metal by colloid particles plays an important part in the all-sliming process and may be cited as one reason at least for the non-recovery of all the extracted metal. It is no longer possible, in the light of contemporary investigations, to speak definitely of dissolved or undissolved metal after cyanide treatment, when it is impossible to determine how much of the metal has been dissolved and absorbed, and how much has been untouched by the solvent. Reduction in actual recovery as a result of the absorption of metal-bearing solution is a phase of treatment not encountered in amalgamation practice. Absorption and adsorption are phenomena which are largely influenced by the question of fineness of division; and this fact may serve to explain the reason for the comparatively high residual content after milling and fine grinding in a cyanide solution carrying valuable metal, or in which a proportion of the metal is immediately dissolved.

**Milling Silver Ores in Cyanide Solution.**—In the case of silver ores the advantages are in favour of milling in cyanide and the abolition of amalgamation, other than in exceptional cases, such as at Nipissing. The reason is that on normal-grade ores amalgamation is an inefficient process at best, a circumstance largely due to the fact that the specific gravity of the metal is comparatively low, and also that it invariably occurs in combinations unfavourable to high extractions by this process.

An additional factor influencing the decision to abolish amalgamation in favour of milling in cyanide with silver ores is found in the circumstance that the finest grinding is necessary in any case, whereas with gold ores it is almost invariably unnecessary, although often adopted to eliminate the need for separate treatments of sand and slime and to simplify the subsequent methods by handling the pulp in one solution and solution-displacement operation.

**Water Consumption and Milling in Cyanide.**—On the question of water consumption there are advantages in favour of milling in cyanide in those

cases where efficient thickening is out of the question, or where the residue is discharged at under 20 per cent. moisture. Under normal conditions, and by inexpensive means, sand tailing may be dewatered to carry the same percentage of moisture as when being dumped. Slime pulp can usually be dewatered by gravitational settlement and thickening to from 25 to 35 per cent. moisture, or from 15 to 25 per cent. by means of continuous-operation vacuum filters. The residue from decantation or vacuum-filter plants usually carries a fairly high percentage of moisture, so that there need be no accumulation of solution, and in most cases there is sufficient balance for a water-wash.

Filter-press dewatering is practised at the Homestake, Great Fingall, and elsewhere, and illustrates the most exact precision of chemical manipulation. The ore is first amalgamated to reduce coarse-gold content and the subsequent cost of chemical treatment by cyanide. It is thoroughly washed during milling, and lime is added previous to cyaniding to counteract soluble refractories and to assist coagulation. Cyanide treatment is then carried out in the presses on a neutralised slime, and unnecessary exposure to atmospheric influences is avoided, the time of cyanide treatment being limited to a minimum. The result is an extraction and recovery unparalleled in the metallurgy of gold ore and a cyanide consumption well below the average.

With ordinary chamber filter-presses a low percentage of moisture is left in the residue, from 15 to 25 per cent. ; but where opportunity permits an extensive water-wash it is often advisable to dispense with its use. It has been found that the practice necessitates a much higher cyanide strength than is necessary for the effective solution of the gold ; and the loss of cyanide when an appreciable percentage of the moisture in the residue is in the form of barren solution is less than when an attempt is made to retain the whole of the solution, plus the contained double salts, in the circuit. In other words, a reduction in cyanide follows the practice of keeping the zinc content in the solution as low as possible ; and the simplest way to ensure this is to discharge a small percentage of the solution with the residue as moisture.

With regard to the use of a replacement water-wash with a filter-cake there are few instances where this is practicable. In most cases the time may be better employed in extending the solution-wash. Water will only mechanically displace dissolved gold, but the use of solution may cause residual gold to dissolve as well as replacing gold already in solution. There is also no danger of upsetting the chemical equilibrium of the solution by dilution.

When the residue is being discharged at a high percentage of moisture, a final and efficient water-wash is, of course, imperative. Under normal conditions, however, the retention of practically all the solution in a closed circuit, with the constant addition of zinc and other impurities, will eventually result in an increased chemical consumption, and a decrease in actual recovery.

**Acidity and Alkalinity.**—A great number of ores have an acid reaction. This acidity should be neutralised before the ore is brought into contact

with cyanide solution, more especially if the latter contains gold. This neutralisation can be effectively carried out by the adoption of milling in water. If milling in cyanide is practised there is a loss of solvent from this cause alone, which is by no means insignificant when circumstances prevent a high alkalinity being maintained in the solution.

On the question of the degree of alkalinity: it is common knowledge that, in the great majority of cases, the amount of electrolyte needed for effective settlement involves a higher alkalinity than is good for satisfactory and economical precipitation. In other cases a very low alkalinity is required for effective solution of the metal in cyanide. When milling and thickening in cyanide solution is practised it is generally found that the question of alkalinity for solution and precipitation of the gold must be subordinate to the degree required for effective settlement of the ore. Under these conditions, and especially in the case of an ore carrying a variable clay or schistose content, it is almost impossible properly to control the alkalinity to suit metallurgical requirements without interfering with the mechanical capacity of the settling plant. On the other hand, when milling in water is practised, a fairly high alkalinity can usually be maintained in the milling circuit with its attendant advantages of improved amalgamation, more effective settlement of the slime, and a clearer overflow for battery-water service. At the same time a thicker discharge can be obtained from the dewaterers. On account of the lower water percentage in the thickened sludge, the resultant alkalinity of the pulp, after having been thinned with plant solution, is so low that alkalinity control is then an easy matter, and can be adjusted to a nicety to suit metallurgical and chemical requirements.

These remarks with reference to latent acidity in the ore also apply to the numerous instances where the ore contains reducing agents, raw or partly oxidised pyrites, or other compounds whose detrimental effects on a cyanide salt, or on a solution of metals of uncertain stability, would be wholly or partly neutralised by the preliminary treatment with lime or other alkali. In the case where the available water is acid or naturally impure, the cost of preliminary treatment may often be avoided by milling in lime water; and a number of the troubles encountered from such a source may be overcome by the addition of sufficient alkali in the battery.

**Volume of Metal-Bearing Solution.**—A further disadvantage of milling in cyanide lies in the fact that the necessary quantity of metal-bearing solution must be very largely increased, and accurate control rendered more difficult. The metal in the battery solution can only be reduced by an increase in the tonnage precipitated, involving further consumption of cyanide and zinc. If the practice of milling an acid ore in a cyanide solution carrying an appreciable quantity of metal is to be avoided, then all the solution coming from the mill should be precipitated. When sliming follows milling it is often difficult, if not unwise, to keep the dilution below 1:10; and the latter figure multiplied by the tonnage would possibly underestimate the amount which ought to be precipitated. In general practice,

however, a compromise is usually effected. A part of the solution is precipitated, and the possible ill-effects of bringing the remainder of the dissolved metal in contact with raw ore is not taken into account. The result is in favour of premature precipitation, colloidal absorption of dissolved gold, and a decrease in the net recovery in the plant.

The loss of cyanide due to the necessity of bringing moisture in the crushed ore up to general average cyanide strength has been estimated on the Rand to amount to 0.12 lb. (KCN) per ton of ore. But as the total cyanide consumption on this field, where milling in water is almost entirely practised, amounts to 0.3 lb. per ton, it is obvious that there are economical advantages in favour of milling in water which greatly outweigh the loss of cyanide from the above cause.

The restricted limit to cyanide operations as a result of milling in water is doubtless responsible for the very low average cyanide consumption; and recent experiments have shown that this could be halved by minimising the contact of cyanide solution with the air, i.e. by closing-in the plant.

**Chemical and Mechanical Loss of Cyanide.**—In milling-in-cyanide practice the exposure of large tonnages of solution to deleterious atmospheric effects, accompanied as it is in many cases by violent agitation in milling apparatus, is also responsible for considerable chemical loss of cyanide due to the evolution of hydrocyanic acid—a gas whose presence is not difficult to detect in many plants.

As regards chemical loss of cyanide J. W. Hutchinson<sup>1</sup> stated that, with reference to Goldfield Consolidated results, a marked increase was indicated by adopting milling in cyanide, the conclusion being confirmed by a three-weeks' test. W. P. Lass also stated,<sup>2</sup> in connection with the preliminary work with Alaska Treadwell ore, that previous agitation with alkali saved 25 per cent. of the cyanide consumption. E. M. Hamilton has shown that,<sup>3</sup> in treating a concentrated pyritic material, direct cyanidation caused a loss of 100 lb. of the solvent per ton, an amount subsequently halved by preliminary treatment with lime water.

Mechanical loss of cyanide when milling in water is practised is largely influenced by the effectiveness of the dewatering equipment. On the other hand, an almost complete displacement of the cyanide solution from the residue may result in fouling, which necessitates a higher cyanide strength in the solution and a greater cyanide consumption than would otherwise occur.

As a matter of fact some misconception exists with regard to this question. Complete displacement of solution by water or wash is seldom or never practised, and in the great majority of instances is impossible to obtain. The ultimate amount of solution discharged with the residue is, more or less, in inverse ratio to the time spent in water washing. In other

<sup>1</sup> *Eng. and Min. Jour.*, vol. xciv. p. 170.

<sup>2</sup> *Min. and Sci. Press*, Oct. 21, 1911.

<sup>3</sup> *Eng. and Min. Jour.*, vol. xciii. p. 840.

words: if it takes an hour to displace rich solution in a thick cake with barren solution, it will also take an hour to displace the barren solution with water. A five- or ten-minute water wash usually suffices, so that the operator of a leaf vacuum filter is probably discharging more solution with his residue after a water wash than a filter-press operator would discharge after solution washing only. If it is admitted that a certain percentage of solution is invariably discharged with the residue then it is obvious why fouling of solution is more in evidence when milling in cyanide is practised. With milling in water, the solution is more easily controlled, and the "barren," with the highest zinc content, is used almost exclusively for washing after filtration. Hence the largest amount of zinc is sent to the dump in the smallest amount of solution. Where milling in cyanide is practised a large proportion of the "barren," containing the highest zinc content, is sent back to the main circuit and distributed over a large bulk of diluent.

The statements which are made with regard to loss of cyanide as a result of milling in water are not borne out in practice, and the majority of operators whose experience has included both methods will probably agree that the advantages of a preliminary alkaline treatment by milling in water, the close control of the solution on account of the restricted amount, and the low tonnage of solution to be precipitated, are all favourable factors ensuring low cyanide consumption in the treatment of a gold ore.

**Time of Treatment after Milling in Cyanide.**—As regards time of treatment both Hutchinson and Lass, already quoted, make decisive statements that, in dealing with their respective ores, a longer time was required to effect solution when milling in cyanide was practised. These results coincide with those obtained by the author, and a number of tests resulted in the conclusion that the gold in mill solution, when brought in contact with fresh ore in a milling plant, was not of so stable a nature as is generally supposed. The tests indicated a great difference when the ore was cyanided after a preliminary alkaline treatment, as compared to a similar handling of the residue from the milling-in-cyanide plant after complete treatment. In the latter case further solution from a comparatively high residue only occurred after protracted periods of agitation and then only in minute amounts, the rate of extraction being unaffected by replacement of solution. On the other hand, when the original ore was agitated direct after a preliminary alkaline treatment, a lower residue was obtained in a very much shorter time.

As facilities for mechanical contact between ore and solvent are far more favourable when cyaniding is commenced in the mill it is probable that this apparent delay in the dissolution of the gold is really due to the fact that absorption, adsorption, and possibly reprecipitation of the metal accompanies its solution by cyanide. It may be assumed that the final "extraction" point is delayed until the reprecipitated or adsorbed gold has been redissolved; and also until as much as possible of the absorbed metal-bearing solution has been osmotically removed by continual contact with a solution of different strength. These final phases would be slow-acting



processes, and the theory may account for the apparent delay in the extraction of gold from ore after it has been milled in a cyanide solution containing gold.

**Gold Recovery after Milling in Cyanide.**—With regard to the question of gold recovery percentage by both methods Hutchinson stated that, at Goldfield, crushing in alkaline water followed by cyanide treatment resulted in an increase of 2 per cent. in the extraction, as against milling in cyanide. On the other hand, and dealing with the case of an amalgamable gold ore, the loss in actual recovery would be proportionate to the amount of gold which could have been extracted by amalgamation.

The Alaska Treadwell tests showed equal extractions with straight cyaniding as compared with cyaniding preceded by amalgamation; from which it may be deduced that practical methods would favour the inclusion of the latter process if net recovery only was taken into consideration. The final conclusion, however, is generally dependent on a number of other factors which might deflect the decision in favour of milling in cyanide.

**Sampling after Milling in Cyanide.**—In the matter of sampling: any decision will be overwhelmingly in favour of milling in water, for the efficiency of the operation is proportionate to the fineness to which the ore has been ground before the sample is taken. This is borne out in assay practice, the assay charge from the sample being the logical equivalent of a sample from a day's run.

There are, of course, mills where an elaborate system of sampling is installed, and where a large proportion of the tonnage is handled and re-ground in the sampling plant. Under such conditions and with an ore carrying a coarse or variable metal content a good average sample may be obtained, but its representative value is seldom comparable with a sample of the whole product that is taken automatically after milling in water. The necessity for the efficient sampling of the ore in a mill cannot be over-estimated, and a deal of unnecessary worry and not a little disappointment would be avoided if more attention could be paid to this detail in the first instance. Nothing is more detrimental to the efficiency of operations than the knowledge that no sure conclusions can be arrived at as to the results of the work being carried on, or any reliable forecast made from the assay results of samples of the ore.

**The Personal Factor.**—Modern milling practice requires constant attention to details, such as the testing of the pulp at every stage of the reduction process. One deterrent factor in cyanide mills is seen in the trouble caused to operators by the poisonous effects of the solution in circuit. In deciding on a process by a comparison of figures the personal factor is often overlooked; and, unless the millmen selected happen to be in the enviable minority and immune from infection, a serious factor against efficient working is encountered by the adoption of milling in cyanide. Infection may result in symptoms varying from slight rash annoyance to perpetual irritation by day and insomnia by night; and the superintendent must often withhold censure for neglect

of duty when he knows that vigilance on the part of the millman would have resulted in increased personal discomfort.

The outstanding features of milling in cyanide may be summarised as follows:

*Advantages.*—1. The milling and regrinding plant offers exceptional advantages for the solution of gold and silver by reason of the intimate contact between metal and solvent.

2. A saving in water consumption may be effected by milling in cyanide in cases where the settlement of the ore is difficult; where the dewatering units are ineffective; or where the residue is being discharged at under 20 per cent. moisture.

3. A simplification of plant arrangement is possible and separate return-pumping and storage systems for solution and water are unnecessary.

*Disadvantages.*—1. Decrease in amalgamation efficiency where this recovery method is practised; corrosion of plates; and increased consumption of mercury.

2. Higher cyanide consumption and longer time required to effect a lower extraction of the metal in the ore.

3. Increase in the amount of metal-bearing solution in circuit, with greater liability for mechanical loss; and proportionately higher precipitation costs per ton milled.

4. Liability of neglect in milling and grinding plant operation on account of the poisonous nature of the diluent.

*Applicability.*—1. When no advantage, either economical, metallurgical, or chemical, arises from the use of a preliminary alkaline treatment of the ore or the water used for milling purposes.

2. When amalgamation, if adopted, would result in an inappreciable saving.

3. When it is not necessary to fine-grind or slime the ore preparatory to cyanidation, thus obtaining a product which, in residue form, usually contains more moisture than if coarser crushing, followed by leaching, had been adopted.

4. When dewatering prior to cyaniding, to within a few percentages of the amount of moisture to be discharged in the residue, is impracticable.

Probably no other phase of any chemico-metallurgical process is being operated under so many different conditions as the one dealt with in this chapter; and for this reason an effort has been made to avoid dogmatic decisions. The necessity for the marshalling of facts and the classification of ideas on this important development is obvious. The subject of milling in cyanide has received scant attention; and the method has often been adopted as a necessary concomitant in the utilisation of a specialised process of solution recovery, and sometimes without an apparent realisation of the advantages in favour of the alternative method of milling and thickening in alkaline water as a preliminary to cyanide treatment.

## CHAPTER XXIII.

### CYANIDE TREATMENT OF DRY-CRUSHED ORE.

IN this application of the cyanide process, the first trouble met with results from the necessity of drying all ore which contains more than about 2 per cent. of moisture, before it can be crushed dry. Drying is done in kilns or furnaces, and this often partly decomposes pyrite and other sulphides, arsenides, and tellurides, forming compounds that increase the consumption of cyanide, and render the gold and silver less soluble.

In the earliest New Zealand practice the ore was dried in circular pits sunk in the ground, each about 20 ft. in diameter and capable of holding 100 tons of ore and 25 tons of firewood, the ore and fuel being placed in alternating layers. The bottom of the pit was formed by a brick arch with a discharge door over a tunnel, through which the dried ore was removed in trucks, in lots of 50 tons at a time. After each withdrawal of half the charge the kiln was filled up again with layers of ore and wood, so that the method was practically continuous.

At the Crown mine at Karangahake, when dry crushing was practised there, the ore was dried in a furnace having a firebox, over which is a steeply inclined rectangular flue provided with inclined iron shelves, so arranged that they prevented the ore from touching the back and front walls of the furnace, and allowed space for the products of combustion to pass up around the lumps of ore: similar in principle to the well-known Stetefeldt furnace. The Rothwell furnace was used for drying in America. It consists essentially of a tapered revolving cylinder, usually about 20 ft. long, 4 ft. in diameter at the receiving end, and 5 ft. at the delivery end. There is a brick firebox at the large end, and a brick dust chamber and chimney at the small end. The cylinder, of  $\frac{1}{2}$ -in. sheet steel, is divided into quadrants by two longitudinal partitions of  $\frac{3}{8}$ -in. steel plates, which cross each other at right angles and extend nearly the full length of the cylinder.<sup>1</sup> The motion and inclination of the cylinder cause the ore to pass slowly along towards the firebox end, and it is dried by the heated gases passing in the opposite direction. Many of the well-known types of roasting furnaces are also used for drying ore.

With most ores the amount of metallic mineral is so great that merely drying the ore renders the subsequent treatment difficult, if not impracticable, on account of the partial decomposition already referred to, and this is especially the case with ores containing much arsenical pyrite and certain

<sup>1</sup> *Engineering*, July 1, 1898.

tellurides. In these cases when the ore is dried before crushing, it is often advisable to roast after crushing, and when roasting becomes necessary, it is usually found most economical to dead roast.

**Direct Treatment of Dry-crushed Ore.**—The method of treating the ore direct as it comes from the dry crusher is used in some localities where the characteristics of the ore favour its adoption. The mixed product is conveyed to suitable leaching vats, and treated entire without preliminary classification. The extraction is seldom high, but this is generally counterbalanced by the fact that the cost of preliminary handling is small when compared with other methods.

An interesting example of American practice by this method is described by H. A. Megraw<sup>1</sup> and refers to the Wasp No. 2 Mill, South Dakota. In this instance the ore is dry crushed in rolls to  $\frac{1}{4}$  inch. The coarse and fine product is stored in bins, and afterwards conveyed to the leaching vats where it is cyanided direct. The extraction averages 70 per cent., but the cost of treatment is only 68 cents per ton. The original gold content of the ore is valued at about \$2.00, and a profit of from 50 to 75 cents is made for every ton of ore treated.

**New Zealand Practice.**—The practice of dry-crushing with stamps has now given place to wet-crushing. Early dry-crushing practice with stamp mills is of historical interest, and the following details are therefore given:

The ore when crushed was usually carried by screw conveyors to storage bins, from whence it was delivered in trucks to the treatment tanks. Sometimes the section of tram line over each vat had a lateral traversing motion, so that a truck upon it could be tipped over any part of the vat. In some plants a complete system of belt conveyors was used for charging. The entire treatment was given in one vat, the vats being only filled to a depth of 2 ft. 6 in. on account of the presence of a quantity of fine material.

It is well known that when ore is treated dry, a larger quantity of wash water can be used than in the case of wet-crushed ore, without increasing the bulk of solution in use. In this district the following figures represent the average practice: For each ton of ore, 0.35 ton strong solution, 0.3 ton of weak solution, and 0.35 ton of water wash.

**Chilean practice with Ball Mills.**—The following account of a large installation of ball mills at the Atacama Mineral Co., Chile, is condensed from a paper read by Sidney H. Loram before the American Institution of Mining Engineers.<sup>2</sup>

The ore is broken in a Blake crusher to pass a 3-in. ring and is then delivered to eight No. 4 Grusonwerk ball mills, each of which contains 1320 lb. of balls and requires 11 h.p. at 25 r.p.m. The output has reached  $6\frac{1}{2}$  tons in 24 hours, but over a long period, including stoppages and repairs, has only averaged  $5\frac{1}{2}$  tons per mill per day when crushing through an 80-mesh screen. Usually 60 per cent. of the product obtained under these conditions will pass a 100-mesh screen. One man can hand-feed the eight mills. The

<sup>1</sup> *Details of Cyanide Practice*, p. 51.

<sup>2</sup> *Trans.*, vol. xxix. pp. 497-502.

amount of steel abraded from balls and plates is 4.27 lb. per ton of ore; in addition to which, 1.42 lb. is thrown away in worn-out parts, so that the total loss amounts to 5.69 lb. of steel per ton of ore crushed. These figures refer to the special malleable steel supplied by the makers of the machines.

The fine grinding is necessary because the gold is extremely fine, and is said to be so thoroughly distributed through the gangue that even minute particles of the latter may sometimes completely enclose the gold. The ore, after crushing, is elevated in trucks to an overhead tram line and tipped into the leaching vats, where it receives an ordinary single treatment with the following quantities of solution per ton:

|                  |                     |                     |        |   |   |
|------------------|---------------------|---------------------|--------|---|---|
| Alkaline wash,   | 0.22 ton containing | 0.01 per cent. KCN. |        |   |   |
| Strong solution, | 0.44 „              | „                   | 0.30 „ | „ | „ |
| Weak „           | 0.22 „              | „                   | 0.1 „  | „ | „ |
| Water wash,      | 0.11 „              | „                   | 0.01 „ | „ | „ |

**West Australian Practice.**—The advantages derived from giving a dead roast before the cyanide treatment of the sulpho-telluride ores met with in the lower levels of the Kalgoorlie mines led, in many instances, to the introduction of dry crushing.

The following particulars of practice are abstracted from Robert Allen's *West Australian Metallurgical Practice*:

**Kalgurli Mine.**—The ore is coarse broken, milled in Krupp ball mills, and roasted in Edwards furnaces. The roasted ore is mixed with cyanide solution and the pulp classified. The coarser portion is reground to slime in Wheeler pans, and then joins the overflow from the classifiers. The slime pulp is thickened and then agitated with cyanide in special air agitation vats, and subsequently filter-pressed. The gold-bearing solution is handled in the usual way.

**Great Boulder Proprietary Mine.**—The ore, after coarse breaking, is milled in Griffin mills and Krupp ball mills. The whole product is then roasted in Edwards and Merton furnaces, afterwards being pulped with cyanide solution. The coarser portion of the pulp is then reground in pans, and the slime pulp subsequently thickened and agitated by mechanical means. The solution is separated from the ore by means of filter-presses and vacuum-filters and the gold recovered in the usual way.

**South Kalgurli Mine.**—The ore, as in all other instances mentioned, is 'all-roasted' and afterwards 'all-slimed.' The material, after rock-breaking, is fed into Krupp ball mills, of which an unusually large size, No. 8, are used. Merton furnaces are used for roasting. Sliming is done in pans, and the thickened slime product is agitated by mechanical means and subsequently filter-pressed. The solution is precipitated by ordinary methods. The extraction for the first three months of 1906 amounted to 94.2 per cent. The gold content in residue amounted to 0.521 dwt. per ton milled.

**Moistening Dry-Crushed Ore.**—A special apparatus has been devised and used for moistening dry-crushed ore, known as Blaisdell's mixer, and is shown

in Fig. 41 in plan and sectional elevation. Its object is to prepare the ore in order that it may be conveyed and distributed in the leaching vats without

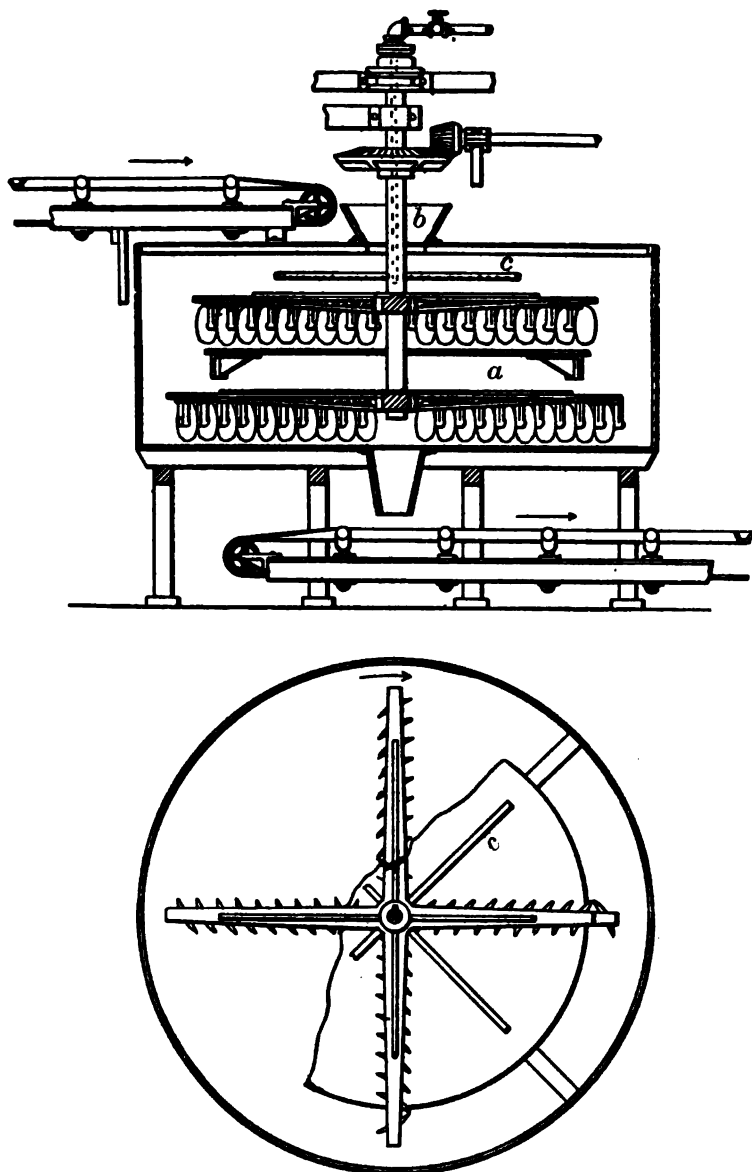


FIG. 41.—The Blaisdell Mixer for moistening pulverised ore with cyanide solution ; sectional elevation and plan.

danger of loss from dusting, and further, to improve the texture of the ore for leaching purposes and aerate it while moistened with cyanide solution. It

consists essentially of two sets of arms, keyed on a vertical suspended shaft, which carry a number of discs, arranged at angles similar to those in the Blaisdell excavator. The upper arms are caused to rotate within a covered circular chamber over a shelf *a*, and the lower ones over the bottom of the chamber. The ore is fed through the hopper *b* and falls on to the shelf *a*, where it gets turned over and over again by the rotating discs, and at the same time is sprinkled with a cyanide solution. This is done by a perforated pipe *c*, which is fed through a hollow in the shaft. The upper set of discs are arranged to throw the ore outwards, mixing it at the same time, when it falls over the edge on to the bottom of the chamber, where the lower discs push it inwards to a central discharge. It then falls on to a belt conveyor to be carried to the leaching vats. A 12-ft. machine is said to have a capacity of 100 to 200 tons per hour, and requires 5 horse power.

## CHAPTER XXIV.

### HEATING AND ROASTING OF ORES.

INTERESTING data have been published<sup>1</sup> on the results, from the mechanical point of view, of heating ore preparatory to crushing. A number of tests demonstrated that heating (to about 800° C.) makes the ore easier to crush; and that heating and quenching leads to a still greater friability. The principle is neither new nor unrecognised, but the figures given are informative. By comparing the outputs of a standard crushing machine with raw and treated material, respectively, it was found that, with three classes of Cornish tin ores, a preliminary heating and quenching treatment resulted in an increase in output varying from 65 to 313 per cent. It was also demonstrated that the effect of heating, and still more of heating and quenching, is to yield a crushed product with a greater proportion of middle-sized particles and a smaller proportion of fine slime. Roasting or heating an ore prior to crushing or grinding has seldom or never been attempted other than in the case where the elimination of some interfering element or compound was desirable. From the mechanical point of view, however, there are several features of interest. Roasted ore may be ground finer than raw ore and still be in excellent condition for leaching treatment. The heating or roasting tends to change the physical characteristics of the colloidal portion of the ore, with the result that absorption is considerably decreased, and the final displacement of dissolved metal is made with greater facility. A second advantage of roasting is the reduction in weight of material to be crushed.<sup>2</sup>

One of the most appreciated features about the cyanide process is that roasting is generally unnecessary. There are, however, a few exceptions, and various attempts have been made to bring even these in line with the more general rule. The efforts made have not been completely successful; so that roasting, in spite of its many drawbacks, cannot advantageously, in certain cases, be dispensed with.

An ore that is suitably roasted to chlorinate is generally suitable to cyanide. The most perfect condition of a roasted ore to cyanide is that in which no deoxidising compounds nor cyanicides exist. The presence of the latter in small quantities is of less importance than the former, because the cyanides simply destroy a certain amount of cyanide or alkali which can be

<sup>1</sup> "Effect of Heating, and Heating and Quenching, Cornish Tin Ores before Crushing," by Arthur Yates, *Trans. Inst. Min. and Met.*, 1918.

<sup>2</sup> A. W. Allen, *Eng. and Min. Jour.*, Jan. 11, 1919.



easily detected, whereas the former prevents the dissolution of the gold and silver in a reasonable time without being recognised until too late.

**Objects of Roasting.**—What is aimed at is to convert the base metals into insoluble compounds, or into compounds which, if soluble, do not act as reducing agents in the solution, and to free the particles of gold and silver from all encasing matter.

An object of less importance to strive for is that, if the compounds formed are soluble, they do not decompose the cyanide solution.

When iron pyrite,  $\text{FeS}_2$ , is roasted the aim is to oxidise it to ferric oxide,  $\text{Fe}_2\text{O}_3$ —a compound which is not acted on by a cyanide solution, and which, if in contact with the metallic gold and silver, assists dissolution. But should some  $\text{FeS}_2$ , or a partly oxidised product, be left in the roasted ore, it dissolves more or less, resulting in a solution possessing a slightly reducing action which to some extent retards dissolution of the gold. If a more soluble sulphide be left in the finished product, such as sodium or calcium sulphide, the rate of dissolution of the gold and silver may be considerably retarded, depending on the proportion of sulphide dissolved.

Then, again, if tellurium is present in the ore, the aim is to drive off as large a proportion as practicable, and thoroughly oxidise what remains to  $\text{TeO}_2$ . The presence of this compound in small quantities has comparatively little effect on the rate of dissolution of the gold. Should, however, some Te escape oxidation, or become only partly oxidised, it also dissolves, and produces a solution which has a strong deoxidising action.

Oxy-compounds are often formed in the furnace, which act as cyanicides, such as  $\text{FeSO}_4$ . This compound not only destroys cyanide, but robs the solution of dissolved oxygen. It does not, however, form a solution with a reducing action.  $\text{TeO}_2$  is injurious by combining with the alkali of the cyanide, leaving  $\text{HCN}$  free in solution—a compound which is practically of no value in the dissolving of the gold.

These compounds, although undesirable, are less injurious than those which form deoxidising solutions, such as sulphides. They destroy a certain amount of cyanide, which is often not of material importance.

The base metallic mineral particles become broken up during roasting into a powder, or are changed into particles of a porous nature. This action, which is more or less equivalent to fine grinding, liberates the gold and silver so that the cyanide solution is capable of acting on them. Should the ore, however, be passed along the furnace bed too rapidly or at too high a temperature, some of the pyrite or tellurides are apt to fuse, forming a matte with the gold and silver, which is difficult to oxidise; and it often happens that the particles of matte escape when the finished product is oxidised only superficially. The gold and silver in these particles are practically encased, so that, in order to attack them by the solution, it becomes necessary to fine grind the whole of the ore after leaving the furnace. This grinding brings the precious metals in contact with the solution, but it has the disadvantage of rendering the matte more soluble, which results in a solution having a

deoxidising action and therefore a low dissolving efficiency on the gold and silver. Consequently, longer treatment may be necessary than when the ore is perfectly roasted, or high residues result.

**Temperature in Roasting.**—The main points to be attended to are that at the feed end of the furnace the temperature is sufficiently high to begin to decompose the metallic minerals, usually  $570^{\circ}$  to  $650^{\circ}$  F., and at the finishing end high enough to oxidise completely the products in a reasonable time without fusion, even superficially, of the gold and silver particles. Temperature may vary not only with the minerals present, but also with the rate at which the ore is rabbled. The brisker the rabbling, which should be continuous and uniform, the higher may be the temperature. As a rule, however, when roasting to cyanide, it is better not to risk the danger attending the highest temperatures the ore will stand, but rather to use a moderate heat, and to extend the time that the ore is kept on the finishing hearth. By this means the danger of leaving reducing agents, such as sulphides, in the finished product is diminished, but the danger of leaving cyanicides, especially  $\text{FeSO}_4$ , may be increased. This latter is, however, the less serious evil.

With certain ores, in order to get a "dead sweet" roast, it is necessary to have a fairly high finishing heat, which is apt to fuse or soften some of the gold particles, not necessarily forming spheroidal globules, but superficially softening them, so that two particles weld together. When this happens, the amount of gold dissolved in the time usually allowed becomes small. When the gold is fused to spheroidal globules, a minimum amount dissolves. Among the great secrets of good roasting are steadiness of temperature, which is not always easy to ensure, together with a uniform rabbling.

**Value of Air in Roasting.**—In roasting to cyanide the most essential factor is a superabundance of heated oxygen. It is conducive to the production of a finished material of a non-reducing character in the cyanide solution, and, although it may also be conducive to the formation of cyanicides, it is nevertheless the safest policy to adopt.

Every oxidisable particle of ore should be brought into actual contact with the oxidising atmosphere. This is laborious work when performed by hand, and is rarely done efficiently by other than mechanical means. Any particles kept below the surface of the ore bed seldom become oxidised so as to be in a fit state for cyaniding; and in the early stages of roasting they are apt to fuse, forming a matte, which afterwards, even in the presence of abundance of oxygen, is difficult to roast thoroughly. The presence of particles of matte accounts to some extent for poor extractions because they dissolve and make the solution act as a deoxidiser. The elimination of substances such as tellurium, arsenic, or antimony need not necessarily be complete for a good extraction, provided that what remains is thoroughly oxidised. The presence of sulphates of alkalis, or alkaline earths, has little or no effect, and even small quantities of sulphate of iron need not be dreaded. On the other hand, minute quantities of the more soluble sulphides, tellurides, and antimonides may have a detrimental effect on the extraction. It therefore behoves the

furnace man to keep these at a minimum by using abundance of air, even at the risk of sulphating.

As the flue gases pass towards the rear of the furnace the percentage of free oxygen present decreases, and the percentage of non-oxidising or even reducing gases increases, until, as is constantly the case, the oxidising effect on the last few feet of the furnace bed is practically *nil*. This state may be brought about by the presence of 10 per cent. of sulphur dioxide, or a smaller percentage of distilled products, such as sulphur or tellurium. The difficulty of getting an adequate supply of oxygen in all parts of the furnace is recognised by the fact that, if the flow be rapid, excessive loss from dusting may occur, or the flue gases may be cooled so as to retard, instead of to increase, oxidation. The rate at which the gases move through the furnace must therefore be governed by the temperature and by the liability of the ore to dust.

**Theory of Roasting.**—It has been pointed out by several authorities that chemical change in the furnace, whereby  $\text{Fe}_2\text{O}_3$  and  $\text{SO}_2$  are formed, is not truly expressed by any single equation, but rather that the transformation takes place in a series of steps, each step requiring a definite period of time for completion. It is generally thought that one atom of the sulphur in the molecule  $\text{FeS}_2$  is weakly held, and, in the roasting process, is oxidised or volatilised before decomposition of the resulting molecule begins. On this subject A. W. Warwick<sup>1</sup> writes: " $\text{FeS}_2$  (pyrite) is a definite chemical molecule, with one atom of sulphur very strongly held and one atom more weakly linked, but not so loosely held as commonly taught. In an oxidising atmosphere at a temperature of  $575^\circ$  to  $600^\circ$  F., the molecule breaks up, possibly hastened by the flying off of the weakly held atom of sulphur. The heat developed by the oxidation raises the temperature to about  $850^\circ$  F. At the period of disintegration of the molecular structure, and while the whole system is in a state of unstable equilibrium, in an oxidising atmosphere, the atoms of Fe and S are completely converted into oxides, and the molecule  $\text{FeS}$  is either never formed or is so transitory that it may, for all practical purposes, be assumed never to exist. . . . The sulphur existing in the roasted product may be accounted for by the amount of pyrites left undecomposed; in no case can  $\text{FeS}$  be found in the calcined ore."

However, if  $\text{FeS}_2$  is imperfectly roasted, easily soluble substances remain, which render the cyanide solution of a more deoxidising character than if the  $\text{FeS}_2$  were treated in its original natural state; so great may this be that the extraction on a badly roasted ore may be less than half of that obtained when the same ore is treated raw.

It is generally recognised that when a portion of the  $\text{SO}_2$  produced, together with the oxygen gas present, come in contact with incandescent silica or other oxides, the latter act as catalytic agents, forming  $\text{SO}_3$ , thus:



In presence of  $\text{FeO}$ , formed in one of the steps during roasting, the  $\text{SO}_3$

<sup>1</sup> *Min. Mag.*, vol. xii. pp. 196-205, 1905.

combines to form  $\text{FeSO}_4$ . This accounts for the presence of sulphates in the roasted product.

**The Reaction Velocity.**—The velocity-value of a chemical change is greatest at the beginning of the reaction, and becomes less and less as the

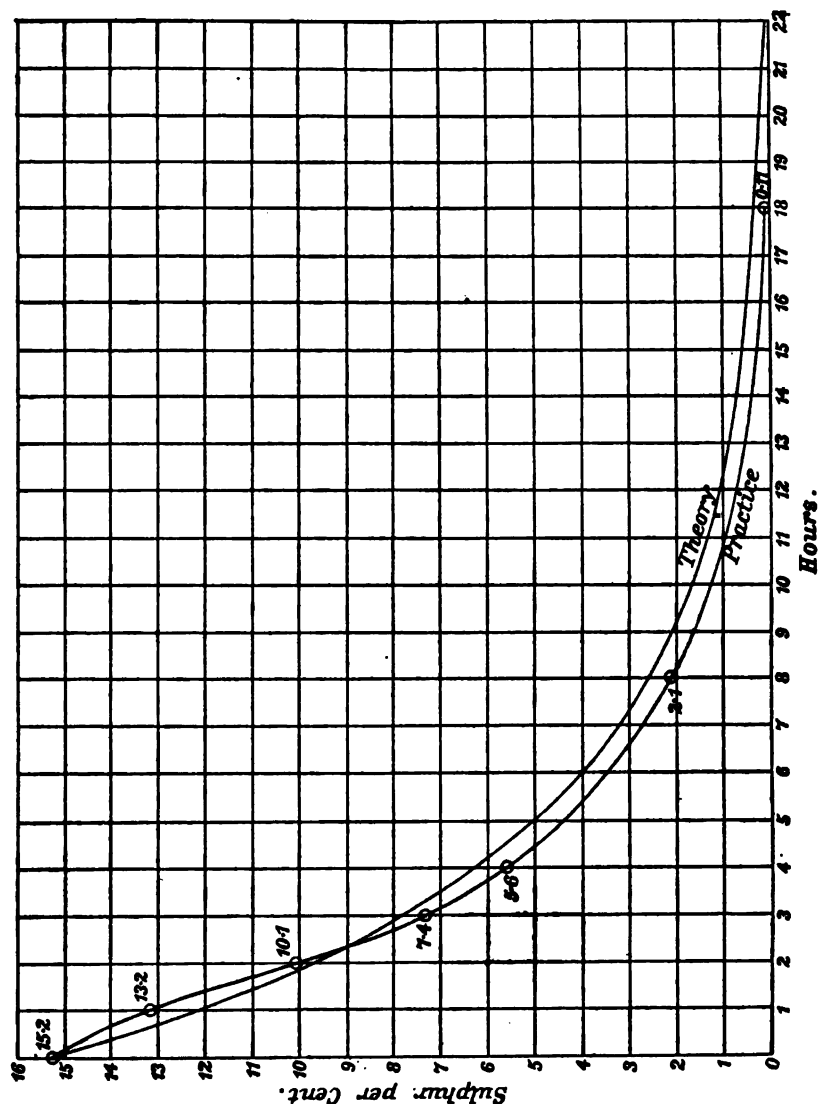


FIG. 42. — Curves showing Loss of Sulphur during Roasting, in Theory and Practice.

reaction proceeds towards completion. It has been practically demonstrated by Warwick,<sup>1</sup> that, in a general way, roasting obeys the same law. His results are given in Table XXVIII., and are plotted in Fig. 42. The third column shows the percentage of sulphur present at various intervals of time.

<sup>1</sup> *Min. Mag.*, vol. xii. pp. 196-205, 1905.

The practical results show that one-third of the sulphur was eliminated in the first two hours, and on this basis the amounts of sulphur that should be eliminated according to the law of chemical velocity have been calculated, i.e. that one-third of the sulphur remaining at the end of each two hours is removed during each succeeding two hours. The results are plotted beside those obtained from practical work, and show a close similarity. The theoretical results obtained assume a constant temperature; whereas in practical work a gradual increase of temperature occurs as the roasting proceeds, which has the effect of increasing the velocity of the reaction.

TABLE XXVIII.—*Showing the Rates at which Roasting Proceeds in Practice and Theory.*

| Time.                 | Temperature. | Practice,<br>Sulphur per cent. | Theory,<br>Sulphur per cent. |
|-----------------------|--------------|--------------------------------|------------------------------|
| Began to burn . . . . | 300° C.      | 15.2                           | 15.2                         |
| 1 hour after . . . .  | 510          | 13.2                           | 12.16                        |
| 2   "   . . . .       | ...          | 10.1                           | 9.73                         |
| 3   "   . . . .       | ...          | 7.4                            | 7.79                         |
| 4   "   . . . .       | 610          | 5.6                            | 6.23                         |
| 8   "   . . . .       | 675          | 2.1                            | 2.55                         |
| 18   "   . . . .      | 750          | 0.11                           | 0.274                        |
| 21   "   . . . .      | ...          | ...                            | 0.106                        |

**The Caron Process.**—An important adaptation of roasting is seen in the new Caron process for the treatment of manganese-silver ores. The  $\text{MnO}_2$  is converted into  $\text{MnO}$  by means of a reducing roast in a revolving kiln, and cyanidation is then found to be economically practicable. In the case of the low-grade manganese silver ore of the Buffalo-Hunter Mining, Milling and Development Co., less than 15 per cent. of the silver can be extracted by direct cyanide treatment, though after the reducing roast the amount reaches 85 per cent.<sup>1</sup>

**Details of Roasting Furnaces.**—When it was found necessary to roast certain ores before they could be satisfactorily treated, many different types of furnaces were tried. Of these, practically only one remains in use, that is, the type with rabblers rotated in circles over a stationary hearth. The Merton and Edwards furnaces, which belong to this type, are now almost exclusively used in W. Australia.

**The Merton Furnace.**—This furnace, as employed with the more recent modifications for roasting to cyanide, is shown in Fig. 43. It has three or more horizontal hearths, superimposed, each about 20 ft. long by 7 ft. wide, on which the roasting is brought nearly to completion. Immediately behind the fire is a finishing hearth, where a dead-sweet roast is produced before the ore leaves the furnace. The finishing hearth shown in the cut is one of the extended type, which is preferred in W. Australia, as it gives a sweeter roast than the former short-hearth type and permits increased output.

<sup>1</sup> A. W. Allen, *Eng. and Min. Jour.*, Jan. 11, 1919.

Four vertical shafts, square in section, pass through the superimposed hearths, and are supported beneath on foot steps, which, for convenience

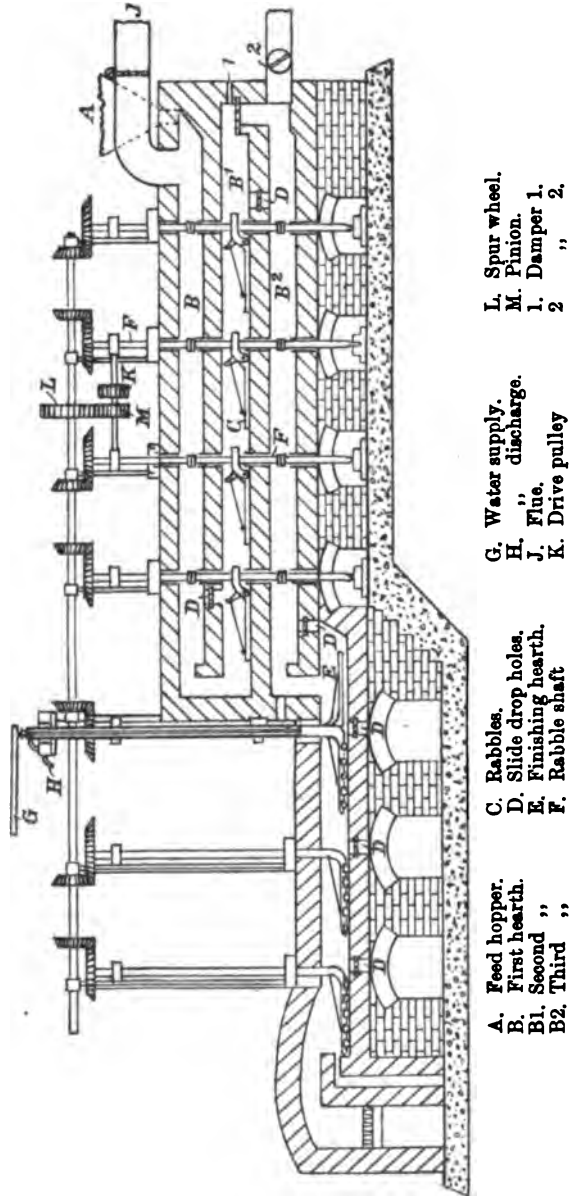


FIG. 48.—Merton's Roasting Furnace: Longitudinal Section.

in adjustment, consist sometimes of miniature jacks. Each shaft rotates in a direction opposite to the one next, and carries three rabbles, which sweep just clear of the hearths. The shafts are a little further apart than the radius of the circle described by the rabble, so that the paths overlap. The result

is that the one rabble pushes a portion of the ore on to the path of the next, which in turn pushes a like portion on to a third, and so on to the end of the furnace.

The same operation gently turns the ore over and exposes it to the oxidising gases. Near the end of each hearth is a slot through which the ore is discharged on to the next hearth. This is fitted with a sliding bar, which is capable of being adjusted to suit the time that it is necessary to keep the ore on the hearth. The finishing hearth is also provided with one or more similar slots, which regulate the rate of the final discharge. The rabble shafts for the finishing hearth are sometimes water-cooled, to increase their durability. This is not absolutely necessary, and is sometimes dispensed with where water is scarce.

The shafts are driven by bevel or worm gearing at the rate of 1 to 2 r.p.m. When worm gearing is employed it is encased in dust-proof boxes and run in oil. Special provision is made to allow the shafts to expand or contract without interfering with the smooth running of the gear.

About 2 h.p. is required when roasting pyritic concentrate, but when dealing with ordinary crushed quartz, only about 1 h.p. is consumed by the rabbling. When the feed to the furnace is constant, for which an automatic arrangement is provided, the capacity is about 20 to 25 tons for ores containing up to 6 per cent. of sulphur, and 5 to 12 tons for concentrate containing up to 25 per cent. of sulphur, per 24 hours. If much lime is present the capacity may be materially diminished.

At the ends and sides of the furnace are doors which may be used to admit air. Those along the sides are conveniently situated for removing and renewing the rabbles. This may be done without lowering the temperature, the operation occupying only two minutes to each rabble.

When the furnace has been regulated to suit the class of ore under treatment, a uniform product may be obtained, containing under 0.05 per cent. of sulphur as sulphide, even when the original contained as much as 6 per cent. To obtain such a result constantly necessitates, however, an unvarying regular temperature, which with wood or coal fuel is difficult to ensure. On the other hand, if the fuel is oil or gas, little trouble is experienced in controlling the temperature and maintaining it constant.

As the roasting finishes, the product is continuously discharged through the slot holes in the hearth, from which it is conducted to a push conveyor that transmits it to a storage bin or cooling floor. While the hot product is being pushed forward by the conveyor, fresh surfaces are continuously exposed to the oxidising action of the atmosphere. This materially helps in diminishing any traces of reducing agents which may have escaped oxidation in the furnace.

The falling of the ore from one hearth to another through the current of oxidising gases is conducive to an effective roast, but an objection is raised to this in dealing with ores containing much dust. The draught may sometimes carry away such a large proportion of the finer particles that the

advantage may be completely discounted. To overcome this objection, the draught passages at the end of the hearths are blocked up, and others are made near the rabble shaft. The discharge slot hole is then inserted between the draught passage and the end of the hearth, so that the ore may fall through comparatively still gases. Also, a sloping plate is placed beneath the slot hole, which conducts the ore still further from the draught and causes it to slide gently down to the hearth beneath.

One man can attend to five or six furnaces burning wood or coal and to a larger number burning oil or gas.

In the construction of the furnace, 10,000 to 15,000 common red bricks, 3000 to 5000 firebricks, and a number of tiles are required. Around the fire and finishing hearth, where the heat is greatest, the bricks should be set in fired clay, but in the cooler parts ordinary brickclay is good enough. About 15 barrels of fireclay are sufficient. The ironwork supplied by the makers weighs about 13 long tons.

**The Edwards Furnaces.**—These are built either fixed on the ground or encased by an iron frame which is supported on a fulcrum near the middle. This enables the hearth to be tilted so as to give it any desired slope. The latter is a popular type, especially where the character of the ore is liable to change.

The furnace consists of two stiffened girders of boiler plate, about 63 ft. long, built on the cantilever principle and placed about 9 ft. apart. These are secured together at the bottom by stout corrugated iron and at the ends by plane iron plate. The top edges are stayed with tie rods. The bottom and sides are lined with brick, and the top is arched across with brick in the ordinary manner. The hearth in the clear is about 58 ft. by 6 ft. 5 in. Fifteen vertical shafts which carry the rabbles pass through the arch and are driven overhead from a single shaft by bevel gearing. The driving mechanism is supported by an iron framework secured to the shell of the furnace. The first 13 rabbles, from the feed end, make about 1 r.p.m., the fourteenth about two, and the fifteenth, which discharges the finished product, four. The object of increasing the speed of the rabbles at the fire end is to give the ore a brisker stirring at the hottest parts, which helps in preventing fusion of any of the metallic particles. The three rabbles nearest the fire are water-cooled.

The whole furnace rests on two pivots at points near the middle of each girder, situated at a height of 2 ft. above ground. It is kept rigid by a small jack at one end, which is also used to adjust the slope. The rabbles push the ore forward in identically the same manner as in the Merton furnace, but the rate of movement is capable of being regulated by the slope given to the hearth, which can be altered at any time in a few minutes. About two to five hours is the time taken for the ore to pass from the feeder to the discharge, and the slope of the hearth is from 3 in. to 1 ft. in the length of 60 ft. The depth of the ore on the hearth varies from  $1\frac{1}{2}$  to 3 in.

The Edwards furnace is capable of producing a finished product quite as



good for cyaniding as that delivered by the Merton, but its capacity is less and the consumption of fuel is greater, which results in higher roasting costs.

It is advisable to construct a furnace of such a capacity that it can be operated continuously, for all furnaces used intermittently are liable to damage, due to expansion and contraction. The Edwards tilting furnace is less liable to injury from this cause than most others, and is especially suitable for small works where roasting is carried on only at intervals.

A tilting furnace of the dimensions given will satisfactorily roast about 16 to 21 tons of W. Australian ore, containing up to 6 per cent. of sulphur, per 24 hours. The labour and power required are about the same as for the Merton furnace.

About 6000 common red bricks, 3000 firebricks, and 10 barrels of fireclay should be provided for this furnace. The ironwork, as supplied by the makers, weighs 19 long tons, and the greatest single piece about 22 cwt.

On large mines, where the ore is of a fairly uniform character, the Edwards furnace, of large dimensions, has been built on the ground in the ordinary way. The best slope for the hearth has to be found by trial, and this is conveniently determined by first operating with a tilting furnace. A double row of rabbles, suitable for wide hearths, is said to give better results than the above-described single row.

It has been found in practice that in order to get a sufficiently gradual rise in temperature, the length of the furnace must not be less than 60 ft. In Western Australia from 60 to 70 ft. is usual; and in American practice, lengths of from 80 to 100 ft. are common for straight reverberatory furnaces.

The widths generally range from 7 to 12 ft., and for structural reasons it is not advisable to exceed 14 ft. for this type.

## CHAPTER XXV.

### THE CLASSIFICATION OF ORE PULP.

CLASSIFICATION, with regard to cyanidation, may be defined as that preliminary to the process which ensures a grade of material suitable to the method of treatment. Efficient classification serves a number of purposes. It may separate a ground ore into two or more products each of which is best treated by a separate method. Thus a concentrate may need an extensive treatment, a sandy product may require leaching for a number of days, and the separated slime may be profitably cyanided and dumped within a few hours. In another instance the function of classification may be to return to the grinding plant a product which is too coarse for cyaniding; or it may serve the purpose of isolating an uncyanidable or unprofitable material.

In the early application of the cyanide process the only available method of treatment was by leaching. It was necessary to treat as much as possible of the tonnage milled by this method and only the crudest classification was practised. Seams of slime invariably occurred in the leaching vats. These interfered with efficient percolation and favoured the retention of the dissolved gold and silver in the residue.

The success of modern leaching methods is largely the outcome of efficient classification. By the removal of practically the whole of the slime a product remains which leaches with such freedom that it is now found practicable to grind the sand to a much finer degree of comminution than was previously thought possible, and so expose fresh surfaces of the metal to the solvent action of the cyanide solution. Thus a satisfactory extraction is often obtained at the lowest cost for both comminution and cyanide treatment.

A further advantage of efficient classification is seen in the adoption of preliminary coarse crushing in batteries. Such practice is only feasible when followed by efficient classification of the coarser product, which is reground in machines better adapted than the stamp mill for such a purpose. By the provision of a separate regrinding plant a selective action takes place during classification whereby not only the larger but also the heavier particles of the ground ore are retained in the regrinding circuit. Hence classification is often synonymous with concentration. The concentratable portion of the ore is retained until it is ground to a finer degree than the bulk of the ore. The advantage of this selective regrinding in the matter of subse-

quent cyanide treatment of the ore is obvious, and forms one of the chief reasons for the success, in so many instances, of leaching treatment which has been preceded by efficient classification.

It is not the intention in the present edition to deal with classifying apparatus to any greater extent than in former editions. The reason for this lies in the fact that whereas classification is an important preliminary to cyanide treatment, it cannot be considered an integral part of the process.

**Classification by Cones and Mechanical Means.**—When separate sand and slime treatment is practised, cones are largely used for classifying and dewatering, prior to regrinding the coarser product; and also for classifying the final pulp and dewatering the sandy underflow. They have been extensively adopted in South Africa, but in the United States and Mexico they have generally been superseded by classifiers of the mechanical type which are capable of definite division into two products in one operation.

**Classification by Shaking Tables.**—Shaking tables of the Wilfley type are in use in various localities as classifiers. They have the advantage over any other type of classifier in that a clean concentrate may be produced, if required, as well as other products. The main disadvantage in their use lies in the fact that their capacity is small when compared with cone or mechanical classifiers.

## CHAPTER XXVI.

### THE CYANIDE TREATMENT OF SAND.

IN the cyaniding of gold and silver ores the larger proportion of tonnage is still being treated by a process involving percolation and leaching. The process of causing a liquid to pass through a porous medium, such as sand, is termed *percolation*; and when the percolating liquid dissolves some of the porous medium it is said to *leach*. 'Leaching treatment' is a phrase applied to describe that method of ore treatment by cyanide solution where dissolution of the contained metals is effected by the gravity percolation of the solution through a classified or unclassified product. This gravity percolation is occasionally assisted by vacuum.

Sliming, followed by filtration, is generally imperative in the case of silver ores; but with simple gold ores, especially those of low grade, the economic applicability of leaching treatment has been amply demonstrated. The percolation method of leaching in vats allows large quantities of material to be dealt with in one operation; the total cost of actual treatment is low; and the plant required is of the simplest description and easy of control.

Modern leaching treatment methods have reached a high state of efficiency as a direct result of the introduction of improvements in classification generally. By careful grading, a battery or tube-mill product may now be freed from practically the whole of its slime content in one operation, the sandy product being delivered from classifiers of the mechanical type with a low percentage of moisture and in a suitable condition for leaching treatment. By cone classification, followed by mechanical dewatering of the underflow product, the same result may be obtained, although a further classification of the overflow is invariably necessary to free it from escaped sand.<sup>1</sup>

The general result of the introduction of efficient classification is that the ores may now be ground much finer than was previously advisable, and the greater proportion of the product resulting may be treated by leaching. A further advantage which has arisen as a direct result of the improvement in classification methods is seen in the abolition of two-stage leaching. The improved extraction which was previously noticeable when a sand charge was turned over or transferred and subjected to an additional treatment is now seldom apparent. The simplicity of the present method and the saving

<sup>1</sup> Vide *Rand Met. Prac.*, vol. i. 212.

in the cost of handling are largely due to the fact that none of the gold in the sand particles is occluded by the presence of slime.

**Rate of Percolation.**—This varies with the volume and uniformity of the interstices between the ore particles, and to some extent with the depth of the column, the pressure, and the temperature. The constituents of the ore also have some influence on the rate of percolation, for minerals such as galena, pyrite, and blende have a marked effect in diminishing the rate of percolation, and this varies with the proportion of these minerals to the siliceous particles present. It naturally follows that with coarser ore particles the larger will be the pores or interstices and the more rapid the rate of percolation. Then, again, uniformity in the size of the ore particles has something to do with the rate of percolation and the leaching effect, for if an ore be crushed to pass through an ordinary assayers' sieve of 90 mesh, and washed free of slime, a good leachable product may be obtained, but if this fine ore be mixed with coarser particles, percolation becomes slower than the average of the two sizes treated separately, and leaching is retarded.

Percolation is carried on in either a downward or upward direction, and Julian made investigations on the rate in both directions. When downward, the rate does not vary very much with different depths of ore, but there is always a diminishing decrease with increase of depth. Thus an ore through which a solution percolated the first foot at an average rate of 1.6 in. per minute, percolated 2 ft. at 1.3 in. per minute, 3 ft. at 1.22, and 4 ft. at 1.2. With upward percolation the average velocity for the first foot from the filter cloth was 8 in. per minute, for 2 ft. was 4 in., for 3 ft. was 2.75 in., and for 4 ft. was 1.4 in. per minute, showing a rapidly diminishing velocity as the solution percolates towards the surface, caused chiefly by the difference of hydrostatic pressure. The level of the solution was in both cases kept constant at 3 in. above the ore surface, for the purpose of comparison.

The loss of actual velocity with increasing depth in the first case is due to the fact that the velocity is proportional to the square root of the head, whereas the resistance to the flow is roughly proportional to the depth of sand traversed. In the second case—upward percolation—the difference is further accentuated because effective head decreases as the depth of sand increases. There are also irregularities in the results which may be due to the presence of globules of air trapped in the solution. In downward percolation the air tends to rise against the current, but with upward flow the air rises with the current, and for shallow depths of sand the air is easily driven out.

**Methods of Applying the Solutions.**—The solution is applied to the ore at the surface and allowed to percolate downwards by gravity, or it is applied to the ore from below the filter cloth and percolates upwards by hydrostatic pressure until it rises to the surface. The rest of the solution is then usually delivered to the top of the ore and is drawn off by gravity.

The solution is applied continuously or intermittently. The former

method consists in adding each solution soon after the previous one has sunk below the surface of the sand, without waiting for the ore to drain. Each succeeding solution is then said to displace the former, and percolation is kept up continuously. By the intermittent method each solution, after it has been allowed to stand for a time, is drawn off, and the ore is drained as well as time will allow, and then the next solution is applied. Some operators apply the solutions in three or four large charges, while others apply them in a good many small charges. Theoretically the latter method should give the best extractions, but in practice the former gives equally good results.

It is in the final treatment, when the water wash is applied, that a number of small charges are highly beneficial. Three or four small charges of 10 tons each will extract more of the salt absorbed by the ore than one large charge of 30 or 40 tons. This is particularly noticeable in the first small charge of water wash, which after percolating will often be found, when drawn off, to be richer in alkali, cyanide, and gold than the previous weak solution.

The addition of a solution of a salt instead of a plain water or dilute cyanide as the final wash would in some cases be beneficial in preventing colloidal precipitates from being broken up and carried through the filter, which so often occurs. The salt causes the precipitated colloids to remain coagulated, and should of course be one that would not react with the cyanide.

The final wash to the ore is one that deserves more attention than is usually given to it; for, after the gold has been dissolved, a considerable proportion often remains behind in solution. This is held chiefly by absorption and capillarity, and the amount removed by each successive wash is a rapidly diminishing quantity. This points to the difficulty of removing the metal even after dissolution, as the number of washes and the quantity of solution capable of being applied in practice is limited by time.

To determine the effect of water washes, Julian saturated a column of clean siliceous sand 100 lb. in weight with 25 gr. of gold as  $\text{KAu}(\text{CN})_2$  in a solution of weak cyanide. He next took a weight of water equal to the weight of sand and divided it into six equal parts. Each sixth was applied separately and allowed to drain completely, and the gold in the sump solutions was determined. Table XXIX. gives the results.

TABLE XXIX.

| No. of Wash. | Gold in Sump Solutions, gr. |
|--------------|-----------------------------|
| 1            | 16.6                        |
| 2            | 4.11                        |
| 3            | 1.8                         |
| 4            | 0.5                         |
| 5            | 0.22                        |
| 6            | 0.07                        |

This shows how rapidly the quantity of gold salt washed out by each wash diminishes, and that after the third wash the amount separated becomes very small. It is problematical whether, with working conditions, it would pay to give the fourth or fifth charge, and the sixth would certainly not be profitable. The gold in solution, in this case, is about ten times as much as would be present in ordinary practice. This is simply to reduce the effect of experimental error.

An identical sample of sand was saturated with gold solution exactly as in the last case and was washed out with water, but in one charge, equal to the weight of the sand, when the total gold separated was 19·8 gr. In the first case the total gold separated was 22·7 gr., showing a better extraction by 2·9 gr. There was, however, in the first case 3·3 gr. and in the second case 5·2 gr. of gold left in the sand. There seems to be room for improvement here, and from experiments made it seems likely that if the final charge of wash waters contained a suitable acid or salt, the last trace of dissolved gold would be completely removed.

The practice of circulating the solution by a small air lift attached to the leaching pipe near the side of the vat was introduced in Western Australia, and was later adopted in some American plants.

**Texture of the Ore.**—Uniformity in texture of the ore is one of the chief requirements for efficient leaching. A correctly classified ore will give more uniform results, a better extraction, and will retain less moisture than the original unclassified product. This is due to the following causes: firstly, that with the sized grains the proportion of voids in the mass is greater than with unsized grains, and consequently there is more room for the passage of solution around the particles, and probably less capillary action to retard its draining; secondly, the conditions being uniform throughout the mass of sized grains, the flow of solution will also be uniform in all parts. Therefore, if continuous leaching be adopted, each part of the solution is more completely displaced by the next following portion, while with intermittent leaching, each separate solution drains off more completely. With mixed sands the moisture retained when drained is generally from 10 to 20 per cent., while with the same grains sized it may be reduced to between 5 and 10 per cent. in the aggregate, according to the perfection of the sizing. Consequently, a smaller quantity of dissolved gold is left behind at each draining, and a higher total extraction is obtained from sized grains with a given amount of washing.

Another advantage is that, in the intermittent system of leaching, the more complete draining and the high proportion of voids allow a more perfect penetration of air between the grains and therefore permit of a more effective supply of oxygen. When grains of sand carrying gold are each coated with solution, and when the whole of this solution surface is in contact with oxygen, all the necessary conditions are present for the rapid dissolution of the gold.

Julian made some determinations as to the volume occupied by the same

weight of ore of different sizes when charged direct with water, moist, and dry, with the results tabulated in Table XXX.

TABLE XXX.

| Passed Mesh. | Retained on Mesh. | Relative volume when |        |      |
|--------------|-------------------|----------------------|--------|------|
|              |                   | Direct.              | Moist. | Dry. |
| 30           | 60                | 100                  | 140    | 96   |
| 60           | 90                | 100                  | 144    | 91   |
| 90           | ...               | 100                  | 152    | 96   |

**The Effect of Slime on Leaching by Percolation.**—Julian procured an ore in which the gold had been very uniformly distributed, so that, when passed through a Clarkson divider, 13 different samples did not vary by assay more than a few grains per ton, and on the average contained 16·6 dwt. gold and a little silver. The ore had, previous to assaying, been washed free from slime, so that when agitated with clean water, and allowed a minute to settle, only a slight turbidity could be seen. Twelve of the samples were mixed with different percentages of slime, varying from 1 to 12 per cent., and each sample was treated by the percolation method with strong solution (0·2 per cent. KCN), weak solution (0·05 per cent.), and a water wash, occupying 72 hours in each case. The slime added was plastic clay, free from gold and silver, containing 89 per cent. silica, the remainder being oxide of iron and alumina. The results are given in Table XXXI.

TABLE XXXI.

Assay of Originals, 16·6 dwt.

| Slime per cent. | Assay of Residues, dwt. Gold. | Extraction per cent. |
|-----------------|-------------------------------|----------------------|
| 0               | 2·6                           | 84·3                 |
| 1               | 2·77                          | 83·3                 |
| 2               | 2·93                          | 82·3                 |
| 3               | 3·4                           | 79·5                 |
| 4               | 3·6                           | 78·3                 |
| 5               | 4                             | 75·9                 |
| 6               | 4·48                          | 73                   |
| 7               | 4·94                          | 70·2                 |
| 8               | 5·5                           | 66·9                 |
| 9               | 5·6                           | 66·3                 |
| 10              | 6·39                          | 62·6                 |
| 11              | 9·3                           | 31·9                 |
| 12              | ...                           | ...                  |



With 12 per cent. slime the percolation became too imperfect to make the results of value.

A portion of the slime had been agitated with cyanide solution, and on testing the solution afterwards it was found to have lost only an inappreciable quantity of cyanide, so that the diminution in the extraction as the slime increased could not have been due alone to decomposition of the solution by the slime constituents.

It might be fairly safe to assume that the diminution in extraction was largely due to the ore particles becoming coated with slime, which hindered the solution from coming in contact with the gold, and also to the difficulty of washing out the dissolved gold, absorbed by the slime.

The loss in velocity, as the solution descends into the ore, seems to be due in a large measure to the pushing forward of the air contained between the particles, and also to the skin friction between the solution and the ore. In its attempt to escape, some of the air moves in an upward direction and sometimes reaches the surface, but a large number of globules are retained, as the pressure does not enable them to force their way upward and escape. These remain in the ore, balanced with the weight of solution above. This has a marked detrimental effect on the rate of leaching, and causes an excess of moisture to be left in the ore after draining. (With upward percolation these globules of air are not formed to the same extent.) Of course, the chief quantity of the air below the ore surface is pushed forward and escapes through the outlet pipe below the filter; and, if the outlet pipe is not open, a large volume of air bubbles upwards through the sand, producing channels and cavities, and affecting the texture of the ore.

**Retention of Moisture.**—The amount of moisture retained by a leached ore depends on the size of the particles, the uniformity of texture, air pressure and temperature, and to some extent on the depth of the column.

Julian made determinations as to the quantity of moisture retained after leaching and draining in the ordinary way. The ore had been washed quite free from slime and was then sized with assayers' sieves. Table XXXII. gives the results.

TABLE XXXII.

| Passed Mesh. | Retained on Mesh. | Moisture per cent. Ore retained. | Remarks.   |
|--------------|-------------------|----------------------------------|--|
| 30           | 60                | 4.5                              | The temperature was between 61° and 65° F. and the pressure below the filter was 1 lb. under the atmosphere. |
| 60           | 90                | 7.4                              |  |
| 90           | ...               | 9.2                              |  |

These experiments were extended by adding various percentages of slime, carefully mixed in a dry state, to an ore which had passed a 30-mesh screen and had been retained on a 60-mesh screen. Table XXXIII. gives the results.

TABLE XXXIII.

| Slime added per cent. | Moisture per cent. Ore retained. | Remarks.   |
|-----------------------|----------------------------------|--|
| 1                     | 8.7                              | The temperature varied between 58° and 64° F. while the experiments were being carried out. The pressure below the filter cloth was not reduced in any case. |
| 2                     | 8.8                              |  |
| 3                     | 10.3                             |  |
| 4                     | 12.5                             |  |
| 5                     | 14.4                             |  |
| 7.5                   | 17.6                             |  |

When the pressure below the filter was reduced by 10 lb. per square inch, the results showed that the moisture retained was irregular, being sometimes greater and sometimes less than the above, but when reduced only 1 or 2 lb. the percolation was very uniform, and the moisture retained was always less than the above. After completing the leaching by gravity alone, the percentage of moisture was always decreased still further, by diminishing the pressure below the filter.

Temperature has some effect, which in many cases is marked. The higher the temperature, the smaller is the quantity of moisture retained.

A deep tank has a slight advantage over a shallow one in the quantity of moisture retained after leaching. There are always a few inches at the bottom of the tank that retain an excessive amount of moisture, and the depth of this, which is usually 2 to 6 in., varies with the nature of the sand, but is independent of the depth of the column. When globules of air or hollows are *in situ* in the tank, the portion immediately above the hollow always retains an excess of moisture, like that at the filter cloth.

**Intermittent Vacuum Leaching.**—The following method of assisting leaching by vacuum has been used in New Zealand: A steel cylinder is connected to the bottom of the leaching vat by means of a pipe provided with a cock. This cock is closed and the cylinder is exhausted by a vacuum pump to about 20 to 24 in. of mercury. The cylinder is then closed to the pump and opened to the vat, so that the vacuum draws the solution into the cylinder. When the latter is full, the cock is closed and the solution is discharged from the cylinder, which is then again exhausted of air. This indirect method of applying the vacuum, in place of connecting the pump directly to the vat, is adopted to prevent injury to the pump valves by grit in the filtered solution.

**Continuous Vacuum Leaching.**—This improved method is used at some of the sand plants on the Rand. At the point where the leaching pipes enter the extractor-house each pipe is connected through a separate cock to a common pipe which leads to a receiver. The receiver, of steel plate, is a vertical cylinder about 12 ft. high by 6 ft. diameter.

Near the bottom of the receiver is an outlet connected to the suction of a three-throw pump which withdraws the solution as it enters the receiver

from the leaching vats, and delivers it to the extractor-boxes. A second outlet, about 4 ft. from the top of the receiver, is connected to the suction of a three-throw air pump which maintains the vacuum. A continuous vacuum leach is, consequently, ensured and the air pump is effectively protected from solid or liquid matter. Some adjustment of the relative speeds of the two pumps is required from time to time according to the quantity of solution arriving at the receiver.

**Influence of Pressure on Percolation.**—Gravity usually supplies the working pressure, assisted by a slight suction from the dip of the outlet pipe. Sometimes, however, a vacuum pump is employed, whereby the pressure below the filter is reduced, and this method is now commonly employed, especially for the final draining of the charge. When the solution is forced through a tank of ore at a pressure near to that of the atmosphere, it is often found that the sand packs and the pores become somewhat choked, whereas if the applied pressure is only slight—about 1 to 3 lb. per sq. in.—this does not occur appreciably. Moreover, in the final draining, the higher pressure tends to leave a larger quantity of moisture behind. This seems to be due to the solution descending too rapidly to allow the globules held between some of the particles to overcome the internal friction, and the air then surrounds these globules and prevents them agglomerating. This is a case of capillarity, and is most apparent in ore loosely charged. If, however, the ore has been drained by gravity alone, the use of a vacuum pump or other suitable appliance at the end of the operation will, as already stated, usually draw off more moisture.

Strong, covered cylindrical tanks have been sometimes employed on ores which were difficult to leach under atmospheric pressure. The solution was pumped in on the top of the ore and a high pressure of air—two or three atmospheres—was applied, which forced the solution uniformly down to the filter bottom. The high pressure causes the solution to become surcharged with oxygen, and thus aids dissolution of the gold. The process is too expensive to be applicable to any but exceptionally rich ores.

**Quantity of Solution to cover the Ore.**—This depends upon the amount of space between the grains, and will also vary according to the nature of the material. The space between the grains fluctuates according to the size of the grains and the uniformity of the grade of material being leached. The quantity of solution required for any particular ore may be determined by finding the weight  $S$  of a certain bulk of the unbroken product, and the weight  $s$  of an equal bulk of the crushed material. Then the proportion of voids, *i.e.* the empty space expressed as a fraction of the whole bulk, is  $\frac{S-s}{S}$ ,

and this space has to be filled with solution in order to cover the ore.

The weight of a cubic foot of ordinary grade leaching product has been found to be 80 lb. The proportion of voids is, therefore, 0.52 cu. ft.; and 1 ton of such sand requires 0.406 ton of solution to cover it.

The volume of solution required in most cases is from  $\frac{1}{3}$  to  $\frac{1}{2}$  of the

volume of the sand; and the weight required is from  $\frac{1}{4}$  to  $\frac{2}{8}$  of the weight of the sand.

**Loss of Solution in Residue.**—In the case of ore crushed in water, the moisture in the sand not only dilutes the first solution applied, but also adds to the total volume of solution in use. To prevent the accumulation of stock solution it is necessary to leave a small quantity of solution in the residue, equal to the original amount brought into the plant as sand moisture *less* a considerable proportion lost in evaporation during treatment. This involves a loss of dissolved gold and cyanide, a disadvantage which is invariably more than counterbalanced in the majority of instances by the fact that the periodic discharging of a small amount of solution daily tends to prevent the accumulation of undesirable zinc products in the solution and obviates "fouling." The loss of gold in residue moisture can be minimised by the use of barren solution for final washes; and the loss of cyanide reduced by the employment of solutions with low cyanide content.

## CHAPTER XXVII.

### THE CYANIDE TREATMENT OF SLIME.

SLIME or slimed ore may be treated either by percolation, by agitation and filtration, or by agitation and decantation.

**Percolation of Slimed Ore with Cyanide Solution.**—Chamber filter-presses are invariably used for this purpose, the practice being to dewater the slime in the press, after the bulk of the gold content has previously been removed by an efficient amalgamation treatment. Cyanide solution is then pumped through the cakes and displaced by ordinary filter-press methods. The time of solution charging varies from 2 hours 20 minutes to 3 hours 45 minutes at the Homestake Mine.<sup>1</sup>

At the Great Fingall Mine, West Australia, the leaching of slime is carried out in ordinary Dehne presses. The time of cyanide treatment is 45 minutes, the raw slime carries from 2.5 to 3.75 dwt. per ton, and the residue from 0.5 to 1 dwt. per ton.<sup>2</sup>

**Settlement of Slimed Ore.**—The separated slime or the ground material, whether from the treatment of gold or silver ore, is thickened in special dewatering vats. These may consist simply of numbers of cones or inverted pyramids so arranged that a continuous or intermittent discharge of the thick sludge takes place from the underflow. There are, however, numerous disadvantages apparent in the working of such apparatus. When the ore carries an appreciable amount of clay, accumulations are liable to form on the sides of the cone which seriously interfere with the efficiency of operation. An additional disadvantage in the use of cone or spitzkasten settlers arises from the fact that continuous operation without a considerable amount of attention is seldom attained. As a result, vertical-sided thickeners, such as the Dorr type, arranged for the mechanical displacement of settled solids, are coming into general use; and their introduction has marked a decided advance in the progress of the handling and treatment of slime, ensuring a continuity of operation and freedom from attention which was unknown before their adoption.

In a few isolated instances slime or slimed ore is dewatered prior to cyanide treatment by means of apparatus involving the use of pressure or vacuum. In this case a much lower percentage of water may be secured in the thickened product.

<sup>1</sup> Clark and Sharwood, *Proc. Inst. Min. and Met.*, Nov. 1912.

<sup>2</sup> R. Allen, *W.A. Met. Prac.*, p. 166.

**Agitation Treatment.**—A certain time of agitation of the slimed pulp is necessary in order that the solution in contact with the ore particles may be continually replaced; and, incidentally, fresh surfaces exposed to the action of the solvent. This proceeding necessarily prevents the settlement of the contained solid and the segregation of the latter from the bulk of the solution. The time of agitation should be as short as possible, and filtration of the pulp should follow as soon as the maximum solution of gold and silver has been recorded.

The thickened pulp as delivered from the settlers or dewaterers is transferred to specially designed agitation vats, being thinned down if necessary by the addition of plant cyanide solution. Two types of apparatus for agitating slime pulp are in general use. One of these relies on the use of compressed air as the motive power to keep the slime particles in suspension and the pulp in efficient agitation. The other system adopts some mechanical means for attaining this end. In some cases a combination of both methods is used. In the use of compressed air for agitation purposes the extraction of gold and silver is hastened by the continuous aeration of the pulp.

**Agitation by Mechanical Means.**—Mechanical agitators are largely used in West Australia for the agitation of slime and slimed ore. The vats are fitted with paddles attached to a central vertical shaft and usually driven by bevel gearing. The speed of the paddles varies with thickness and composition of the pulp. In some instances mechanical agitation of slime has been known to give a more rapid solution of gold and silver than air agitation in Pachuca vats. This may be due to the fact that in the air-lift agitation system the rearrangement of ore particles with reference to solvent is at a minimum in the body of the tank with the use of a normal pulp, and the rate of circulation with the tank in full action exceeds the rate of settlement of solids in the absence of agitation, and operates in the same direction.

It is not improbable that dissolution of the gold and silver occurs in the body of the Pachuca vat, but the main displacement and rearrangement of particles of ore and solution undoubtedly takes place in the lift, and it is upon this that extraction is mainly dependent.

**Agitation by Centrifugal Pumps.**—In several instances centrifugal pumps are used to agitate the pulp. The disadvantages of the system lie in the fact that only a small percentage of the total pulp passes through the pump at one time, so that active agitation of the whole charge is a lengthy operation. Accumulations of slime are also liable to occur in vats unprovided with mechanical stirrers.

The most successful application of centrifugal pumps in the agitation of ore slime is seen on the Rand, where immense pumps are used to transfer the slime charges from one decantation tank to another, the pumps thus acting both as slime agitators and pulp transferers. The success of this method depends upon the fact that the slime treated is mainly a natural slime and contains only a small percentage of slimed quartz. Hence none

of the usual disadvantages of accretion and settlement in the tanks are noticeable. The slime, owing to efficient previous amalgamation treatment of the ore, is exceptionally low grade; and a very limited agitation treatment is necessary.

**Filtration.**—By the term filtration is understood that part of the process whereby the bulk of valuable solution is first removed from the pulp by means of pressure against, or vacuum through, a filtering medium. The residual moisture is subsequently replaced by barren solution, and the latter in some cases by water. Absolutely perfect replacement of solution by wash is seldom possible with any system of filtration, but actual recoveries of from 97 to over 99 per cent. of the dissolved metal are a common occurrence.

Filters in general use may be subdivided under two headings according as to whether pressure or vacuum is used for the separation of ore from valuable solution:<sup>1</sup>

| <i>Pressure Filters.</i>                        | <i>Vacuum Filters.</i>  |
|---|---|
| (i) Chamber type—<br>(a) Dehne.<br>(b) Merrill. | (i) Intermittent—<br>(a) Movable type.<br>(b) Fixed type.       |
| (ii) Cylinder type—<br>(a) Burt.<br>(b) Kelly.  | (ii) Continuous—<br>(a) Drum type.<br>(b) Horizontal leaf type. |

**Preparation of Accumulated Slime.**—On the Witwatersrand, large accumulations of old slime have been dealt with in the following manner:—

The material is hauled up an incline and tipped into wooden vats fitted with horizontal revolving arms, which sometimes carry vertical pins to break up the lumps of slime. The vats are continuously fed with water at the rate of about 2 tons per ton of dry slime, and are provided with outlets at several levels for the continuous discharge of the mixture of slime and water. The trucks of slime should be tipped in at equal intervals to enable the stirring gear to work smoothly.

The lowest discharge outlet, placed about 1 in. or so above the floor of the vat, is only opened for a short period before stopping the gear, in order that the stirrer may work itself quite clear of slime and thus be ready for restarting. The outlet in ordinary use may be about 2 ft. above the vat bottom, and it is convenient to have the stirring arms at such a height that the tops of the pins just show above the pulp when the discharge from this outlet is normal. A relief outlet may be placed about 12 in. higher to prevent the vat from overflowing, in case the ordinary outlet should for any reason be temporarily insufficient.

An inside depth of 4 ft. is sufficient for these pulping vats, and Table XXXIV. gives suitable speeds for the stirring gear and the quantities that can be pulped with various sizes, as well as the approximate power absorbed.

As the slime always contains some sand, which settles on the floor of the pulping vat, a bottom discharge door is provided for shovelling out this deposit

<sup>1</sup> Filtration apparatus is dealt with in a subsequent chapter.

periodically, say once a week. To get rid of any sand which may be discharged with the slime, the outflow from the pulper is usually passed through a small pointed box on its way to the collecting tank, which is used to settle the slime. As accumulated slimes are generally acid, the water overflowing from the collecting vats is preferably run to waste. The settled slime is neutralised with lime and is then ready for transfer to the treatment tanks. In some cases, however, cyanide solution is used in the pulping vats instead of water.

TABLE XXXIV.

| Diameter of Pulping Vat in feet. | Revs. per min. for Stirring Arms. | Tons of Slime pulped per hour. | Approximate h.p. required. |
|----------------------------------|-----------------------------------|--------------------------------|----------------------------|
| 10                               | 16                                | 1.8                            | 2.5                        |
| 12                               | 14                                | 2.6                            | 8                          |
| 14                               | 12.25                             | 3.7                            | 4                          |
| 16                               | 10.75                             | 5.0                            | 5.5                        |
| 18                               | 9.5                               | 6.4                            | 7.5                        |
| 20                               | 8.5                               | 8.0                            | 10                         |

**Decantation Practice on the Rand.**—The slime pulp as received from the classifying plant is delivered into large steel vats from 40 to 70 ft. in diameter, with vertical sides 10 to 12 ft. deep, and with shallow conical bottoms in which the inclination ranges from 1 in 7 in some plants, to 1 in 5 in others. The pulp is delivered into a central intake box or tube which dips down 2 or 3 ft. below the top of the vat. The slime settles as fast as it is received; and, consequently, as soon as the vat is full, clear water overflows into the rim launder. The supply of pulp can be maintained until a sufficient quantity of slime is collected. The contents of the vat are then allowed to settle and the remaining clear water is decanted off through a pipe. This settlement and decantation generally occupy from 8 to 10 hours. Table XXXV. contains average figures showing the relation between the size of vats and quantities of slime handled.

TABLE XXXV.—*Slime-Settling Vats on the Rand.*

| Diam. of Vat, feet. | Assumed Depth at centre, feet. | Tons Dry Slime collected per hour. | Feed per min. of 15.1 pulp, cubic feet. | Total Dry Slime in one charge, tons. |
|---------------------|--------------------------------|------------------------------------|---|--------------------------------------|
| 30                  | 12.5                           | 4.5                                | 37                                      | 70                                   |
| 35                  | 12.9                           | 5.6                                | 46                                      | 100                                  |
| 40                  | 13.3                           | 7.0                                | 58                                      | 135                                  |
| 45                  | 13.8                           | 8.6                                | 71                                      | 175                                  |
| 50                  | 16.2                           | 10.5                               | 86                                      | 255                                  |
| 55                  | 16.6                           | 12.8                               | 105                                     | 315                                  |
| 60                  | 17.0                           | 15.6                               | 128                                     | 385                                  |
| 65                  | 17.4                           | 18.8                               | 155                                     | 465                                  |
| 70                  | 17.8                           | 22.4                               | 184                                     | 550                                  |



The settled slime is sluiced by pressure jets of solution through a central discharge orifice in the vat bottom to the suction pipe of a centrifugal pump which delivers the mixture of slime and solution to the circulating vat. This transfer takes about 4 hours. In this operation about 3 or 4 tons of solution is used per ton of dry slime, including the moisture remaining after the first settlement. The charge is then circulated through the vat for several hours by a centrifugal pump which continuously draws pulp from the bottom and delivers it to the top. After this, the pump transfers the charge to one of the first set of treatment vats.

In some plants Pachuca vats with air agitation have been installed to obviate the circulation with pumps, the latter being then merely used for transferring the charges. In either case circulation and transfer occupy together about 12 hours, and the greater part of the gold is dissolved during these operations. The circulating and treatment vats are similar to the collecting vats, except that the rim launder is omitted.

In the first treatment vat the charge is settled, and as soon as the solution at the top is clear the decanter pipe is lowered to about 1 in. below the surface, to draw off the clear solution; and it is lowered again from time to time as required. The settlement in this vat takes about 52 hours. The slime pulp is then transferred as before to one of the second set of treatment vats, where the processes of settlement and decantation are repeated and together take about 60 hours. Finally the slime is sluiced out of the last vat with water and pumped to the residue dam, this occupying about 4 hours. The total time of treatment, exclusive, is therefore about 144 hours. The solutions used usually contain from 0.015 per cent. to 0.008 per cent. KCN, and the quantity used averages about 7 tons per ton of dry slime.

**Adair-Usher Process.**—The aim of this method is to cause a vertical flow of solution during the settling of the slime in the treatment vat, the upward velocity being equal to the rate at which the slime would settle in still water. The slime is, therefore, held in suspension and leached by the ascending solution. To effect this object the bottom of the vat is provided with a system of radial pipes perforated with  $\frac{3}{16}$ -in. holes, through which barren solution is supplied. This principle of upward displacement of the gold-bearing solution by barren solution is ingenious but impracticable; for, when slime has been pulped with a liquid, each particle tends to hold some of the original liquid and to carry it down through the fresh solution. In experimenting with a slime pulp in small vessels to which a coloured liquid was admitted centrally at the bottom, it has been found that the pulp settled down as a whole, and the coloured liquid rose to the top, each being apparently unchanged.

**Disadvantages of the Decantation Process.**—One objectionable feature of the decantation process is the great dilution of the pulp during treatment, which necessitates large vat capacity and the precipitation of considerable amounts of solution. Another disadvantage is that the slime pulp after natural settlement still contains up to 50 per cent. of moisture.

In the early days of slime treatment, Chas. Butters tried filtration as an aid to settlement and decantation. After each settlement the solution was simultaneously decanted from the top and drawn off through an ordinary filter bottom by a vacuum pump. By this means the moisture is said to have been reduced to 28 per cent. in the residual mass, which remained as a tough leathery layer about 6 to 8 in. in thickness and so coherent that it could be rolled up on the filter.

Table XXXVI. shows the calculated extraction for residual pulp at 28 per cent. moisture. A comparison of these figures with those in Table XXVI. shows that two decantations assisted by filtration give, in all cases, a better extraction than three operations with natural settlement only. With any given quantity of solution, up to seven times the weight of the ore, one operation with filtration gives a better result than two decantations with natural settlement.

It is a realisation of these facts that has led to the extensive adoption of filtration in modern methods of slime treatment.

TABLE XXXVI.—*Extraction obtained by combined Decantation and Filtration.*

| Total Solution<br>per ton<br>of Dry Slime. | Number of Decantations. |       |
|--|-------------------------|-------|
|  | 1                       | 2     |
| 4  | 90                      | 96.69 |
| 5  | 92                      | 97.81 |
| 6  | 93.33                   | 98.44 |
| 7  | 94.29                   | 98.83 |
| 8  | 95                      | 99.09 |

**The Effect of Viscosity of Pulp.**—The first trouble is chiefly a mechanical one, of which Butters<sup>1</sup> says: "One important point in the solution of the gold was not really understood until we had been working on a large scale for a long time; that is, the concentration of the slimes in the liquid, or in other words, the thickness of the pulp. A ton of liquid which is agitated with half a ton of slimes forms a smooth viscous pulp, in which, if you suspend a can of cyanide (which dissolves readily in water), the lumps of cyanide remain for several hours undissolved. In other words, the thick pulp will not readily dissolve even so soluble a substance as cyanide, whereas if you thin the pulp to 4 or 5 ton of liquid to 1 of slime, soluble substances will dissolve very rapidly." The above analogy to the solubility of gold is perhaps not quite an accurate one, but it forcibly presents to the practical mind the effect of thick pulp in diminishing dissolving power of a solvent.

Julian and Smart made some determinations on the solubility of gold in a KCN solution containing slime in different ratios, to show the effect due

<sup>1</sup> *Trans. Chem. and Met. Soc. of S. Africa*, vol. ii. p. 5, Feb. 1898.

to viscosity. These results are given in Table XXXVII. and are plotted in Fig. 44.

The strength of the solution in each case was 0.11 per cent. KCN. The slime was free from organic matter and soluble sulphides, so that the loss in solubility of the gold with increase in thickness of pulp may be considered to be chiefly due to increased viscosity.

TABLE XXXVII.

| Ratio of Solution to<br>1 of Slime by weight. | Rate of Dissolution<br>of the Gold. |
|---|-------------------------------------|
| KCN Sol. alone.                               | 100                                 |
| 6   | 58.6                                |
| 5   | 56                                  |
| 4   | 52.4                                |
| 3   | 47.3                                |
| 2.5   | 44.8                                |
| 2   | 40                                  |
| 1.5   | 34.2                                |
| 1   | 20                                  |

The reason of increased viscosity reducing the solubility would appear to be largely due to a retarding influence on the movements of the K and CN ions. Agitation by stirrers diminishes this influence but little when the pulp is thick, as the mass tends to move as a solid whole rather than to diffuse.

Another source of loss in dissolving power is that the fine solid particles dissociate the KCN molecules and adsorb the potassium, leaving the cyanogen free to combine with water. Now, cyanogen is not a solvent for gold, and when it combines with water it forms HCN and HCNO. The former has but a slight action on gold, and the latter less. When excess of alkali is present, KCN and KCNO results, and this accounts to some extent for the fact that, when dissolving gold from slimes, a cyanide solution always shows a poorer extraction without the addition of alkali or an alkali earth.

**The Uses of Oxygen in Thick Pulp.**—The second difficulty is chiefly a chemical one, due to the presence of substances capable of combining with absorbed oxygen.

Fresh slime includes only the usual products of partly decomposed minerals; but slime, being in a very minute state of subdivision, is much more acted on than the same weight of the same substance in a coarser condition, as found in sand. On the other hand, old slime which has been lying for a considerable period, sometimes many years, in contact with surface water, and often overgrown with vegetation, contains quantities of organic matter that act as reducing agents, and combine with the absorbed oxygen in the solution. Further, it has been pointed out by Caldecott<sup>1</sup> that slime lying in dams containing pyrite undergoes a partial decomposition, with formation of ferrous sulphide (FeS), sulphurous acid (SO<sub>2</sub>), and sulphuretted

<sup>1</sup> *S. African Mining Jour.*, Oct. 23, 1897.

hydrogen ( $H_2S$ ), all of which act as reducers and rob the solution of absorbed oxygen.

Butters<sup>1</sup> says that 6 to 12 hours' agitation is sufficient to dissolve the gold in fresh slimes, whereas 48 hours with accumulated slime failed to do so. This is largely owing to want of oxygen, which becomes used up in oxidising reducing agents. Therefore, in treating old slime, an effective supply of

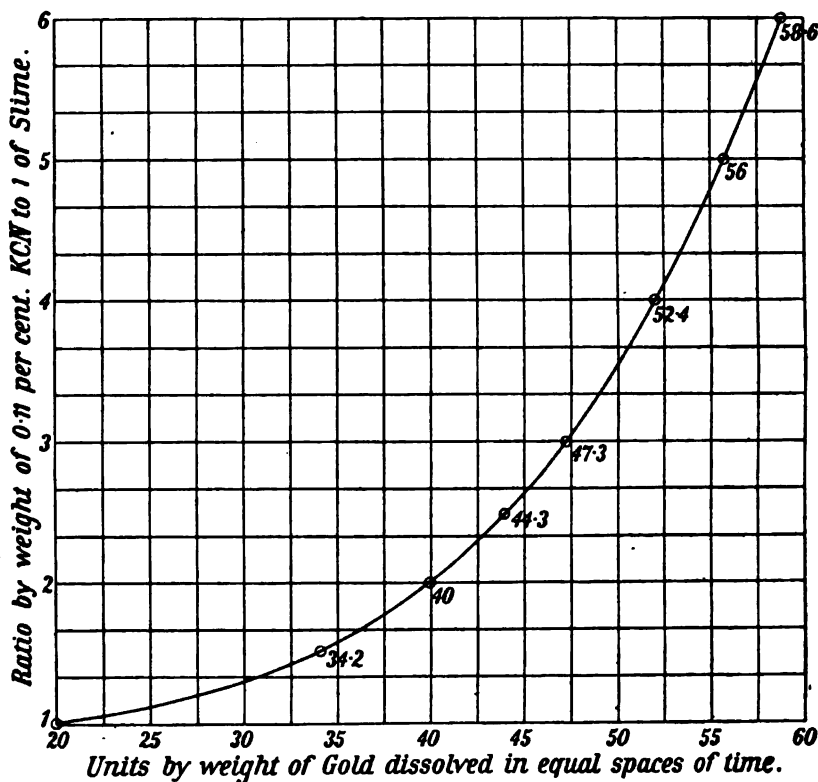


FIG. 44.—The Effect of Viscosity of Pulp on Dissolution of the Gold.

oxygen by aeration or other means is an imperative necessity, and in the treatment of all slime its value can hardly be overrated.

When oxygen is introduced into the pulp, its use is first to oxidise any reducing matter, such as soluble sulphides and organic matter, and as this is effected, the dissolution of the gold and silver takes place. When all the reducers are completely oxidised, the time required to dissolve the gold in true slime is very short, often less than an hour, provided oxygen is supplied as wanted. Owing to this, strong oxidisers may beneficially be added to the pulp, as the gold dissolves completely before they have lost the whole of their oxidising power, whereas if strong oxidisers are applied to coarse particles,

<sup>1</sup> *Jour. Chem. and Met. Soc. S. Africa*, vol. ii. p. 5, 1898,

the time that they are effective is only a fraction of the whole time required to dissolve the gold.

**Velocity of the Reaction.**—When an energetic oxidiser is added to a dense solution, the velocity of the reaction is slower than when added to a dilute solution. Thus, when a strong oxidiser is added to an ordinary clear cyanide solution, the velocity of the reaction may be so rapid that its oxidising effect is many times as fast as demanded by the dissolving metal, and therefore waste goes on; but when the same oxidiser is added to slime pulp, the velocity of the reaction is enormously reduced, so that the oxidising effect may be produced at a rate which approximates that required to dissolve the metal.

**The Oxidation or Aeration of the Pulp.**—Some operators have found it an advantage, when dealing with accumulated slime, to add first to the pulp a considerable quantity of a strong oxidiser, and then run in the cyanide solution. Others have done the oxidising by blowing air through the pulp. Various mechanical means for the aeration of the pulp have also been used.

One of the first methods was to force compressed air into the pulp through a perforated pipe fixed at the bottom of the vat. This was done before and after the addition of the cyanide to the pulp, and has been adopted on a large scale, but was superseded by a cheaper means of aeration introduced by H. T. Durant,<sup>1</sup> namely, the introduction of air into the centrifugal pumps, used to circulate and transfer the slime pulp. He placed a valve in the suction pipe of the pump, and between this valve and the pump inlet he fixed a small air inlet valve. By partly closing the main valve the supply of pulp is restricted to less than the full capacity of the pump, and consequently the latter draws in a small but regular supply of air. This method should, of course, only be applied to a pump fixed below the level of the pulp supply, otherwise the air would interfere with the action of the pump.

**Continuous Decantation Methods of Slime Treatment.**—Attempts were made during the early days of the cyanide process to evolve a method of continuous slime treatment by passing the pulp through a series of conical vats.

The arrangement devised by E. T. Rand will serve to illustrate this method. The overflow from the sand settling vats is thickened by a spitzkasten, or by a conical vat of such a size that the amount of pulp fed would take 3 hours to fill it. The clear water overflow from this goes as usual to the reservoir, the thickened underflow being sent to a dissolving vat with a conical bottom, and on its way to this vat the pulp passes along a launder where KCN is added to it. From the launder a pump delivers the pulp to a Butters distributor at the top of the dissolving vat. The same pump, by a separate pipe, at the same time draws pulp from the lowest point of the vat and delivers it to the same distributor; so that this pump not only supplies pulp to the vat, but also continuously circulates it. A second pump continuously withdraws pulp from the dissolving vat at the same rate as it enters and sends it to a conical vat (having a cone angle of about 60°), which, for convenience, may be termed the first settler. The clear solution overflow

<sup>1</sup> *Jour. Chem. and Met. Soc. S. Africa*, vol. ii. p. 59.

passes to the precipitation boxes, and the thickened underflow, after being again diluted with wash solution, passes into a second conical settler. The clear overflow from the latter is strengthened by the addition of cyanide, and is used to dilute the fresh pulp, which is then on its way to the dissolving vat. The underflow from the second settler carries away the treated slime.

A practical objection to these early methods is the fact that an underflow discharge of thick pulp of uniform dilution is difficult if not impracticable to maintain from a simple cone. Unless the moisture in the underflow can be reduced to 50 per cent. the method is inferior to the ordinary intermittent decantation process.

In 1901, according to Luther B. Eames,<sup>1</sup> a plant was built in the Black Hills of South Dakota by John Randall, employing the same principles. The cones were operated in series, the thick underflow of the first one forming, with a stream of diluting solution, the feed to the second cone of the series. Barren solution was added to the tank immediately preceding the discharge

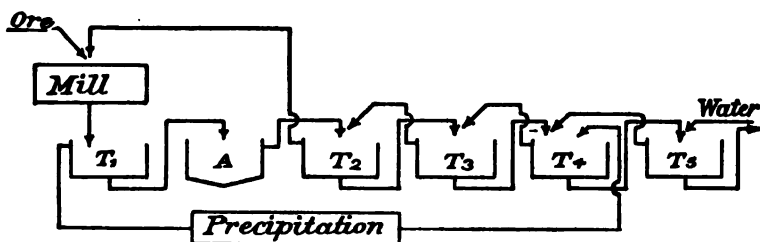


FIG. 45.—Typical Flow Sheet of Countercurrent System.

tank and, after being slightly enriched by the low-grade pulp in this tank, overflowed to form a diluting solution again for the richer feed entering the third tank from the end of the series, and so on back to the richest tank of the series. Clear water was used for the wash in the final tank. This is the principle on which all successful countercurrent decantation plants operate at the present time, but Randall's plant was not successful because of mechanical difficulties in getting a continuous thick discharge from his cone tanks. For a number of years the process was not used, and it was not until the introduction of the Dorr thickener that the minds of metallurgists began to turn again to the continuous decantation principle.

In 1910, two decantation plants were built making use of flow sheets similar to that used by Randall nine years before, but substituting Dorr thickeners for the cones. One of these was at Mocerito in Sinaloa, Mexico, and was installed under the direction of C. Dupre Smith, while the other was designed by J. V. N. Dorr, assisted by Eames, for the Vulture Mines Co. of Wickenburg, Ariz. Both of these pioneer plants were so successful as to encourage further installations.

**Countercurrent Decantation.**—The theory of countercurrent decantation has been summarised by Eames.<sup>2</sup> Fig. 45 indicates a typical flow sheet of

<sup>1</sup> *Trans. Am. Inst. Min. Eng.*, Feb. 1917.

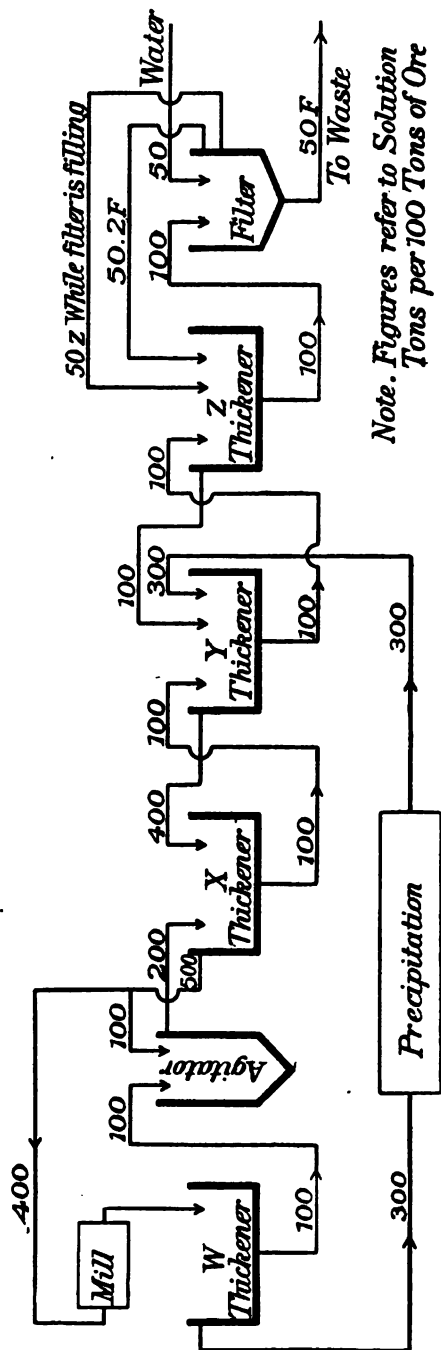
<sup>2</sup> *Loc. cit.*

the countercurrent system,  $T_1$  to  $T_5$  representing the continuous thickeners of

the Dorr type, and *A* an agitator operating on the continuous inflow and outflow method.

This flow sheet assumes that crushing is done in solution, the overflow from the tank  $T_2$  being used for this purpose. This crushing solution leaves the grinding circuit with the ground pulp and enters  $T_1$ , and that part which does not pass to the agitators with the pulp overflows  $T_1$  and goes to precipitation. After depositing its gold contents, it is used to dilute the underflow of  $T_3$  as it enters  $T_4$ . The overflow of  $T_5$  is also mixed into the feed to  $T_4$ . The overflow of  $T_4$  mixes with the underflow of  $T_2$  to form the feed to  $T_3$ , and so forth, as indicated in the flow sheet. At each succeeding mixture the solution meets a pulp of higher dissolved content than itself and is enriched while the pulp is correspondingly impoverished. The pulp at each step approaches the discharge end of the mill while the solution goes to the feed end—hence countercurrent decantation.

**Forecasting Countercurrent Decantation Results.**—The countercurrent system can be designed to operate under a large variety of conditions, and the equipment can be arranged in a number of ways. In the case where the gold or silver is already in solution it is only necessary in many cases to utilise thickeners. In other cases one or more agitators are found necessary. In many instances the advantage of



vacuum filtration of the final discharge is realised. Fig. 46 represents a system of this kind.

The calculation of theoretical estimates is of value for comparison of the various methods of treatment available and in order to ascertain the most suitable system. The method of arriving at such an estimate is shown in the subsequent calculations, which have been abstracted from one of the bulletins issued by the Dorr Company.

#### CALCULATIONS TO ASCERTAIN THEORETICAL DISSOLVED METAL LOSS.

##### Conditions assumed:

- (a) 100 tons of ore per day crushed in cyanide solution.
- (b) Discharge from all thickeners with 50 per cent. moisture.
- (c) 10 units of gold or silver (oz., dwt., or \$) dissolved per ton of ore; 75 per cent. in mill and 25 per cent. in agitators.
- (d) 300 tons of solution from thickener W precipitated to 0.02 unit.
- (e) Agitation with a dilution of 2 of solution to 1 of solids.
- (f) Displacement efficiency of filter, 60 per cent.; that is, 60 per cent. of the metal content of the solution in the solid cake, which is assumed to contain  $33\frac{1}{3}$  per cent. moisture or 50 tons of solution to 100 tons of solids, is recovered. The 50Z returned from the filter to the last thickener represents 50 tons of solution removed in loading the filter, which will, of course, still have the value of Z.
- (g) Let W, X, Y, Z, and F represent units of gold or silver per ton of solution discharged from the thickeners and filter respectively.

##### Equating out of and into each thickener:

- (1)  $100W + 800W = 400X + (.75 \times 10 \times 100)$ .
- (2)  $100X + 500X = 100W + 100X + 400Y + (.25 \times 10 \times 100)$ .
- (3)  $100Y + 400Y = 100X + 400Z$ .
- (4)  $100Z + 400Z = 100Y + 50Z + 50(\frac{1}{3}F) + (300 \times .02)$ .
- (5)  $50F + 50Z + 50(\frac{1}{3}F) = 100Z + 50$  tons of water.

##### Simplifying:

- (1)  $W = X + 1.875$ .
- (2)  $X = Y + 1.0987$ .
- (3)  $Y = Z + .2734$ .
- (4)  $820Z = 38.844$ .
- (5)  $F = .4Z$ .

##### Solving:

- (1)  $W = 3.3464$ .
- (2)  $X = 1.4714$ .
- (3)  $Y = .3776$ .
- (4)  $Z = .047$ .
- (5)  $F = .0417$ .

##### To check these figures:

The amount precipitated from 300 tons @ 3.3464 - .02 = 997.92 units.

The amount lost in tailings 50 tons @ .0417 = 2.08 units.

The amount dissolved: 100 tons @ 10.00 = 1000.00 units.

##### From the foregoing the following results are deduced:

Assay of the pregnant solution, i.e. of W = 3.3464 units.

Assay of the discharged solution, i.e. of F = .0417 units.

Loss of dissolved metal per ton of ore  $\frac{100}{100}F = .0208$  unit.

Dissolved metal saved, 99.8 per cent.

#### CALCULATION FOR MECHANICAL LOSS OF CYANIDE.

##### Conditions assumed:

- (a) Neglect the cyanide consumption throughout the system.
- (b) Strength of cyanide per ton of solution, 4.0 lb.
- (c) Let W, X, Y, Z, and F represent the strength in pounds of cyanide per ton of solution discharged from the thickeners and filter respectively.



Equating out of and into each thickener:

$$(2) 100X + 500X = 100W + 100X + 400Y.$$

$$(3) 100Y + 400Y = 100X + 400Z.$$

$$(4) 100Z + 400Z = 50Z + 50(\frac{1}{3}F) + 300W + 100Y.$$

$$(5) 50F + 50Z + 50(\frac{1}{3}F) = 100Z + 50 \text{ tons of water.}$$

Simplifying:

$$(1) W = 4.0.$$

$$(2) X = .8Y + .8.$$

$$(3) Y = .9524Z + 1905.$$

$$(4) 824.76Z = 1219.05.$$

$$(5) F = .4Z.$$

Solving:

$$(1) W = 4.0.$$

$$(2) X = 8.8124.$$

$$(3) Y = 8.7655.$$

$$(4) Z = 3.7537.$$

$$(5) F = 1.5015.$$

$$\frac{100}{3}F = 7507 \text{ lb.} = \text{Mechanical loss of cyanide per ton of ore.}$$

The system of calculation takes no account of any dissolution of gold or silver that may take place during the countercurrent washing, nor the effect on ultimate recovery of the adsorption or absorption of metal-bearing solution by the particles of ore and colloid slime.

**Agitators and Drum Filters.**—As the reduction of moisture content plays an important part in the countercurrent process it is reasonable to assume that, by using mixing vats and vacuum drum filters, alternately in series, the latter being used merely as dewaterers, a high extraction of dissolved metal could be obtained. The amount of gold or silver present would, of course, determine the extent to which the technical efficiency of such a plant could be carried.

## CHAPTER XXVIII.

### THE CYANIDE TREATMENT OF GRAVITY AND FLOTATION CONCENTRATE.

THE cyanide process is now applied in many cases to materials which, a few years ago, would without question have been treated by roasting and subsequent chlorination. This remark applies especially to the products of close concentration, which are often cyanided raw, but, if rich enough, and contain little or no silver, are preferably roasted. The cyanide process was at first applied to these refractory materials only, which were then treated by agitation, alternating with percolation. From a purely technical point of view, this is undoubtedly the quickest and most efficient method of extracting the gold from such material, whether raw or roasted. But the expense of continuous or repeated agitation of coarse particles of high specific gravity limits the application of this method to rich material only.

R. W. Lodge, of Boston, Mass., published<sup>1</sup> some interesting results of experiments by W. A. Tucker bearing upon this point. The tests were made upon Frue-vanner concentrate, containing about 66·5 per cent. of arsenical pyrites, some iron pyrite, galena, and chalcoppyrite, that assayed over 6 oz. per ton. The solution used contained 1 per cent. of KCN. The results are shown in Table XXXVIII.

The experiments were made on small samples of only 25 gm. each, which explains the evident discrepancies in the KCN consumption, but they nevertheless establish the advantage of agitation for that particular material.

TABLE XXXVIII.

| Percolation.       |                      |                           | Agitation.         |                      |                           |
|--------------------|----------------------|---------------------------|--------------------|----------------------|---------------------------|
| Time of Treatment. | Extraction per cent. | KCN consumed lb. per ton. | Time of Treatment. | Extraction per cent. | KCN consumed lb. per ton. |
| 16 hours.          | 58·69                | 5·16                      | 2 hours.           | 87·28                | 2·48                      |
| 71 „               | 60·45                | 9·84                      | 5½ „               | 91·49                | 4·64                      |
| 118 „              | 76·98                | 9·52                      | 23 „               | 91·90                | 2·96                      |

Similar experiments by John M'Connell on concentrate containing 0·5 oz.

<sup>1</sup> *Trans. Am. Inst. Min. Eng.*, vol. xxv. p. 90.

gold and 13.9 oz. silver, crushed to pass a 90-mesh sieve, give very similar results, thus—

TABLE XXXVIII.

| By Percolation. |                       |                      |             |         | By Agitation.  |                       |                      |             |         |
|-----------------|-----------------------|----------------------|-------------|---------|----------------|-----------------------|----------------------|-------------|---------|
| Time in hours.  | Per cent. KCN before. | Per cent. KCN after. | Extraction. |         | Time in hours. | Per cent. KCN before. | Per cent. KCN after. | Extraction. |         |
|                 |                       |                      | Gold.       | Silver. |                |                       |                      | Gold.       | Silver. |
| 70              | 1.2                   | 1.0                  | 70          | 23.4    | 16             | 1.2                   | 1.1                  | 90          | 83.8    |
| 70              | 0.6                   | 0.45                 | 64          | 12.5    | 16             | 0.6                   | 0.49                 | 82          | 66.5    |

In the case of concentrate from gold ores the refractory elements may often be rendered harmless by a preliminary roast. With silver ores, however, this modification is inadmissible on account of high volatilisation losses.

The advantages of roasting a simple pyritic concentrate obtained from a gold ore were demonstrated at the Sons of Gwalia Mine, Western Australia, where an extraction of 96 per cent. was obtained by cyanidation after roasting as compared with 90 per cent. without roasting.<sup>1</sup> The cyanide consumption is also much less with roasted than with raw concentrate.

The necessity for fine grinding all concentrate as a preliminary to cyanide treatment is generally recognised, although in isolated cases leaching treatment is still practised. In the latter case, however, an extended time of treatment is generally necessary.

With gold ores, the concentrate, whether roasted or not, is generally reground in pans or tube mills if not already of the required fineness. The slime product is treated in an agitation vat which is preferably arranged so that continuous decantation of rich solution and replacement with barren solution takes place concurrently with the agitation of the pulped concentrate. The agitated pulp is then either treated separately in a special filtration plant, is allowed to join the main slime pulp flow to the ore agitation vats, or goes direct to the main ore-filtration department. In either case the subsequent treatment has already been described under the "Cyanide Treatment of Slime." The solution of an economic percentage of gold and silver from concentrate is generally a matter of time, and herein lies the chief reason for the isolation of this product and provision for special agitation treatment.

Robert Linton has described<sup>2</sup> a number of tests made on a silver sulphide concentrate. The results of these tests indicate that cyanide strength should

<sup>1</sup> R. Allen, *West. Aus. Met. Prac.*, p. 44.

<sup>2</sup> *Proc. Chem. Met. and Min. Soc. of S. Africa*, July 1912.

be raised towards the end of treatment, that a high protective alkalinity is unnecessary, and the addition of lead acetate of no material benefit.

The method of treatment of gold ore concentrate at the Goldfields Consolidated mill is unusual, but a high efficiency is obtained by a combination of two separate cyanide treatments, one before and one after roasting. According to J. W. Hutchinson<sup>1</sup> the material is subjected to a preliminary cyanide treatment in order to minimise the loss caused by roasting, which amounts to a  $\frac{1}{2}$  of 1 per cent. of the estimated metal content in the ore. It has been demonstrated that the combination treatment can be given as economically as either of the single treatments and with better results.

**Preliminary Treatment with Acid.**—It often happens that pyritic material, either by oxidising in the mine, or by weathering after concentration, or even through imperfect roasting, may become partly decomposed; and several suggestions have been made with a view to the preparation of such material for cyaniding, without the expense of a complete roast. Some interesting work has been done in this direction by A. F. Crosse and W. Bettel, and from their statements of their respective experimental results<sup>2</sup> the following brief summary is extracted:

Assuming that the free acid and soluble acid salts have been dealt with in the usual manner, the principal cyanicide remaining in the Rand concentrates is the basic ferric sulphate, which, being insoluble in water, is not removed by the ordinary preliminary water wash. Crosse found in one case 0.495 per cent. of iron existing in this condition. His view is that this ferric sulphate is decomposed on adding cyanide solution, forming hydric cyanide and ferric hydrate, and he also states that the freshly precipitated ferric hydrate forms ferro- or ferri-cyanide of potassium. He proposed to overcome the difficulty by dissolving the iron by means of a weak solution of  $H_2SO_4$  or  $HCl$ ; and gave the following example of the results of this treatment:

Concentrate in natural state.—Consumption of KCN, 2.84 gm., and extraction, 51.8 per cent.

Concentrate after digestion with weak  $HCl$ .—Consumption of KCN, 0.66 gm., and extraction, 84.3 per cent.

The advantages of this method are endorsed by W. Bettel, who worked independently on the same lines and arrived at similar results. His views are, however, slightly different as to the action of the ferric sulphate, which, he says, with cyanide of potassium will form ferricyanide of potassium, hydric cyanide, and ferric hydrate, but that the *latter*, when free from basic salts, is entirely unattacked by KCN of moderate strength. Therefore he says, that any method which will completely convert all the basic ferric salts into ferric hydrate and an inert salt, will prevent much loss in cyanide. For instance, sodium dioxide in excess appears to oxidise pyrite to ferric hydrate at once, without forming the intermediate ferrous state, sodium sulphate being

<sup>1</sup> *Min. and Sci. Press*, Jan. 25, 1913.

<sup>2</sup> *Proc. Chem. and Met. Soc. of S. Africa*, vol. i. p. 98 *et seq.*

also formed. He does not, however, propose to use sodium dioxide for this purpose in practical treatment, because the cost of it would be too great.

Crosse also found that by heating the concentrate to a temperature just under dull red, sulphurous acid is given off, and the insoluble ferric sulphate is thereby converted into ferrous sulphate and ferric oxide. It is still necessary to add a little  $H_2SO_4$  to the first two or three water washes, to prevent the formation of the basic ferric sulphate from the ferrous salt in the tank. By this method he obtained 88 per cent. extraction, from concentrate assaying about 3 oz. per ton, in 14 days. One objection to this appears to be the difficulty, when working on a large scale, of exactly regulating the temperature, with the consequent possibility that some of the material might be more refractory after heating than before. However, it is quite possible that some of these experiments might have resulted in practical application had not the method of rough separation of the pyrites from the battery pulp by spitzlutte given such good results.

**Concentrate Treatment in New Zealand.**—At the Waihi Mine the concentrate from tables and vanners is treated in a special plant.<sup>1</sup> This product is mixed with cyanide solution and fed into tube mills. Ten pounds of slaked lime is added per ton of solids. The ground product is elevated to a spitzkasten, the underflow returning to the tube mills. The overflow goes to thickeners, which deliver a pulp containing 0·7 of water to 1 of solid. This pulp is given 8 to 10 days agitation in 23 Pachuca vats, 6 ft. diam. by 16 ft. deep, operated in series. The strength of the solution is maintained at 0·5 per cent. (KCN); and the consumption of NaCN is 16 lb. per ton of concentrate.

The average assay results for one year from material carrying 6 oz. 13 dwt. gold, and 96 oz. 7 dwt. silver, showed a theoretical extraction of 95·46 per cent. of the gold, and 93·27 per cent. of the silver.

The concentrate analysed as follows:

|                      |                 |
|----------------------|-----------------|
| Sulphur, . . . . .   | 35·00 per cent. |
| Iron, . . . . .      | 32·43 „         |
| Zinc, . . . . .      | 2·05 „          |
| Manganese, . . . . . | 1·02 „          |
| Lime, . . . . .      | 0·85 „          |
| Lead, . . . . .      | 0·32 „          |
| Arsenic, . . . . .   | 0·10 „          |
| Copper, . . . . .    | 0·05 „          |
| Silica, . . . . .    | 27·65 „         |
| Loss, . . . . .      | 0·53 „          |

**Treatment of Black Sand.**—This product has been profitably handled at the Geldenhuis Deep, and other mills on the Rand, in the following manner:—<sup>2</sup> The black sand is first carefully separated from the amalgam by being ground

<sup>1</sup> *Eng. and Min. Jour.*, Aug. 5, 1911.

<sup>2</sup> R. Lindsay, *Jour. Chem. Met. and Min. Soc. of S. Africa*, July 1911.

in barrels and in a batea, and subsequently passed over plates. It is then ground in a tube mill until about 90 per cent. passes 200 mesh.

A conical tank, devised by A. F. Crosse, is used for the extraction of the gold by cyanide. This tank contains a small air lift and is so arranged that continuous decantation of solution is possible from a clear zone at the top of the tank. The cone at the Geldenhuis Deep is 7 ft. 6 in. diam. by 8 ft. 4 in. deep, and treats a charge of 1800 lb. of black sand.

The charge is first agitated for 12 hours with about 3 tons of 0.02 per cent. (KCN) solution by means of the air lift; and in this time about 97 per cent. of the gold is dissolved. More solution is then added at the rate of half a ton per hour; the air agitation and the decantation of clear solution being continued concurrently for a period of 12 hours. By this time the gold content is reduced to 4 dwt. per ton. As the original gold content averages 250 dwt. per ton this is equivalent to an extraction of 98.4 per cent., beyond which it is not economical to continue the treatment. The consumption of sodium cyanide is  $2\frac{1}{2}$  lb. per ton of sand; and the consumption of air amounts to 7 cu. ft. per minute.

**Flotation and Cyanidation.**—Nothing can be more absurd than to forecast the passage into oblivion of tried, efficient, and economical processes whenever a new method appears. A number of metallurgists, having announced the demise of amalgamation some time after the introduction of cyanidation, were obliged to busy themselves in writing epitaphs to distract attention from the fact that the older process was still very much alive. The same wild claims were made when flotation became popular, and it was maintained in many quarters that it would displace cyanidation to a large extent. Others went further in presaging a radical modification in modern metallurgical methods when it was realised that amalgamable gold could be floated. Whether the proposal to abolish amalgamation in favour of (1) extracting the gold by an oil process, (2) mixing it at the same time with impurities and gangue, (3) drying the concentrate produced, (4) roasting it, and then (5) cyaniding it—only to get a proportion of the original amalgamable gold content in the form of a more or less base bullion—was ever taken seriously may be doubted. There are, unfortunately, too many engineers who are content to follow blindly the unbalanced predictions that invariably result from the introduction of a new process, regardless of its applicability or comparative worth.

**Oil Flotation of Gold and Silver Ores.**—Two excellent examples of floatable gold and silver ores are found at Guanajuato, Mexico, and Cripple Creek, Colorado, respectively. In the former case flotation was adopted extensively, and proved a welcome addition to or substitute for cyanidation at a time when chemicals were almost unobtainable. The results of adopting flotation, in so far as the net extraction of gold and silver was concerned, varied widely and from day to day. A large proportion of available reserves in Mexico is in the form of stope fillings, and it was found that such material was almost impossible to treat. Moreover, it was demonstrated that it was impracticable

to mix freshly broken ore with "fill" ore and obtain consistent results by subsequent flotation. Only clean material could be treated by the newer process. When the precaution of excluding the remainder was observed the results were, in many instances, excellent.

At one Mexican plant it was found<sup>1</sup> that the average extraction of both metals was higher than the average possible recovery by cyanidation. The extraction of gold was above expectations, but there was a lowering in the extraction of the silver. In this connection it was pointed out that the differentiation between the two products—the bullion recovered by cyanidation, and the concentrate extracted by flotation—is a pertinent distinction. The advantage of securing a final output in the form of bullion is obvious, and the popularity of flotation has been lessened on this account.

In some cases it has been found practicable to treat flotation concentrate on the property, but, speaking generally, the results have been such that little information has been available and the inevitable conclusion has been drawn. At Cobalt, at the famous Nippissing mine plant, the difficulties experienced were insurmountable, and the concentrate from the flotation plant was, for some time, shipped to a smelter. This practice was discontinued early in 1918, however, and gravity concentration was reinstated, together with the practice of cyaniding the concentrate produced.

**Cyaniding Flotation Concentrates.**—The difficulties attending the treatment of concentrates by cyanide have been summarised by Charles Butters<sup>2</sup> who points out that the process of concentration collects in a small bulk, not only the valuable constituents of the ore, but also those substances that act as cyanicides, so that their influence, per ton of material treated, is greater than would be the case with the unconcentrated ore. Heavy minerals, such as the sulphides of iron, copper, lead, arsenic, antimony, zinc; and double sulphides such as mispickel, proustite, pyrrargyrite, and bornite, naturally tend to accumulate in the concentrate. If some interval of time elapses between the formation of the concentrate and its treatment, oxidation may take place, with formation of sulphates, arseniates, and antimonates, which are still more detrimental to cyanide treatment than the original minerals. These difficulties have been wholly or partly overcome by the adoption of modifications in the treatment, such as preliminary water, acid, or alkali washing, roasting, fine grinding, the use of special solvents such as bromocyanide, and prolonged contact of the material with cyanide, extending in some cases to over a month.

In the case of concentrate produced by flotation, Butters continues, the minerals composing the product are substantially the same as those obtained by gravity concentration, and it is to be expected that the same difficulties will be encountered in their treatment. But, as the concentrate also contains a considerable part of the oil, tar, or other flotation agent, the presence of this foreign matter must also be taken into account. A part of

<sup>1</sup> "The Flotation Process," by A. W. Allen, *Eng. and Min. Jour.*, Jan. 11, 1919.

<sup>2</sup> *Min. and Sci. Press*, vol. iii. p. 954.

this is soluble in the cyanide or alkali used in the process, and the solution so formed may be capable of absorbing oxygen. The effect produced by carbonaceous matter in precipitating gold and silver previously dissolved by cyanide is well known, and has been a source of much trouble in many localities. Some of the constituents of this matter are not easily eliminated, and appear to resist oxidation even at a high temperature. Roasting under ordinary conditions does not completely remove the carbon; and it is probable that a portion derived from tar remains in the graphitic form and is then capable of acting as a precipitant for gold or silver.

**Effect of Flotation Oils on Subsequent Cyaniding.**—The effect of flotation oil, *per se*, on cyanide solution was investigated by J. E. Clennell.<sup>1</sup> A 0.24 per cent. KCN solution was used, without the addition of alkali, and experiments made with six different oils. The results were negative so far as cyanide consumption was concerned. This evidence disposes of the contention of P. W. Avery<sup>2</sup> that the consumption of 20 lb. of cyanide per ton in a particular instance of cyaniding flotation concentrate was due to the flotation oil used.

**The Field for Flotation.**—The flotation of an ore, followed by the cyanidation of the concentrate produced, results in three possible sources of loss of metal. There is an unavoidable loss in the tailing from the flotation plant, which may, of course, be cyanided; but if such a procedure is adopted then the question would narrow itself to a comparison of the merits of table *versus* flotation concentration. The second source of loss occurs during the extraction of the metal in the flotation concentrate by means of a cyanide solution, and is usually appreciable. The third loss occurs during the various stages of operation when the extracted metal is being recovered in bullion form. This final loss is, however, usually negligible; and so two main sources are evident. Simple cyanidation, on the other hand, results in only one major loss—that incurred in the residue. These considerations will help to explain why there is a logical (though often unrecognised) reason for the ultimate low recovery of metal by the combination of processes, in many instances where the newer method has been adopted in conjunction with the older.

The feasibility of a compound process—flotation of the ore, followed by cyanidation of the concentrate produced—depends largely on whether a net improvement can be made over existing methods. Under the majority of circumstances it cannot be considered as a rival to straight cyanidation, where the latter method has been found preferable to gravity concentration, followed by cyanidation of the concentrate. The failure is often due to the comparatively low net recovery, which is, in turn, partly due to the fact that successful cyanidation is not restricted to cases where exact adjustment is necessary. With flotation it is different. Not only must adjustment be exact, but failure is liable to result if the class of material going to the mill varies to any appreciable extent, or if non-crystalline or colloidal slime is included.

<sup>1</sup> *Min. and Sci. Press*, vol. cxii. 700.

<sup>2</sup> *Min. and Sci. Press*, vol. cxii. p. 661.



If the extraction of the concentrate in the first place is low, or liable to fluctuate, a factor of uncertainty is introduced which inevitably leads to different results. Further, the flotation concentrate must be prepared for cyanidation by some intermediate step, such as roasting; and difficulties may and do occur in the mechanical handling of such material. Sometimes a high degree of colloidity results in heavy expense for preliminary drying. In those cases it has been demonstrated that, if flotation gives a higher extraction of the mineral and associated gold and silver than gravity concentration, a field may exist for the substitution of the newer process as a preliminary to cyanide treatment.

## CHAPTER XXIX.

### PRECIPITATION BY ZINC.

THIS was the first commercially successful method of precipitating gold and silver from cyanide solutions. The credit is due to MacArthur and Forrest who prepared the zinc in the form of fine thin shavings or turnings. MacArthur has recently<sup>1</sup> described a method of using the zinc in the form of small discs or laminae, and a measure of success seems to have been obtained by the adoption of this modification. It is probable, however, that the method is only suitable in the case of high-grade solutions such as result from the cyanidation of silver-gold ores, and where a high cyanide strength in the solution assists in the formation of a loose precipitate.

As a general rule, zinc as a precipitant is used in one of two forms—either as dust or as shaving. In exceptional cases both varieties may be used to precipitate from the same solution. In this case precipitation by dust is followed by ordinary zinc-shaving precipitation, and a higher-grade dust precipitate is available as a result of the restricted amount of precipitant used. Under normal conditions of working, an excess of dust is added to effect thorough precipitation, and, as no screening-out of the remaining zinc is feasible, the resultant slime is of lower grade than would be the case if less zinc had been used and only partial precipitation ensured.

The introduction of new and improved methods in the application of zinc dust as a precipitant of gold and silver from cyanide solutions has been attended with considerable success. At the same time the use of shavings is still predominant in the industry; and the conservative retention of the older system is doubtless due to the simplicity, if not economy, of its application, coupled with the fact that it is capable of universal application in the cyanidation of nearly all classes and grades of gold and silver ores.

**Method of Use.**—As is well known, the method consists simply in passing the gold and silver solutions through a mass of zinc shavings loosely pressed together, when the gold and silver deposits on the zinc surface. This is usually done in what is termed an extractor box and which is divided, by cross divisions, into a number of compartments (see Figs. 163–167), and so arranged that the solution flows upwards through alternate wide divisions containing the zinc, and downwards through alternate narrow divisions containing only the solution. The zinc rests on loose false bottoms or trays, covered with sieving or perforated plate, which are arranged at a height of

<sup>1</sup> *Proc. Chem. Met. and Min. Soc. of S. Africa*, June 1913.

from 4 to 6 in. from the bottom of the box—the object being to facilitate an even flow of solution, and to leave a clear space where the precipitate can collect without being disturbed.

In some few cases boxes have been made with compartments of the same width and all containing zinc, so that the solution passes continuously through the shavings alternately up and down.

**Preparation of Zinc Shavings.**—Shavings are generally cut as required in a specially designed lathe. This machine has a mandril with a keyway into which the edge of the sheet is inserted and fixed with a key. The sheet is then wound on the mandril, a second sheet is then lapped under the end of the first and wound on also. The roll of metal is tied together by wire twisted around it, or the loose edge is lightly soldered on the roll, and the shavings are turned off the end of the roll by a square-nosed cutting tool, actuated by an automatic feed of the usual kind. A roll or attachment is provided for leading off the shavings from the cutting tool and thereby reducing the necessity of superintendence to a minimum.

In small mills it is seldom advisable to cut the zinc on the property because superintendence costs are much higher when only a small amount is required; and wastages occur, due to lack of skilled attention, which usually more than counterbalance the saving in first cost. Filament zinc of any thickness can be purchased direct from the leading supply firms, and only requires teasing out before being ready for the boxes.

**Thickness and Width of Shavings.**—The thinner the filaments the greater is the active surface for a given weight of zinc. When the filaments are very thin, however, they tend to break up more readily and agglomerate, causing an uneven flow of solution and an excess of zinc in the precipitate. In order to avoid this it is essential that the thickness and width of each filament is uniform throughout its length. In this respect the zinc cut in an automatic lathe is superior to that cut in the hand lathe universally used in early practice.

The thickness of shavings should depend on the conditions prevailing. In the cyanidation of silver ores, and where all short zinc is returned to the boxes, it is customary to use a coarsely cut shaving of about  $\frac{1}{400}$  (0.0025) in. in thickness. In the cyanidation of gold ores, on the other hand, and where the short zinc resulting may be treated with acid at periodic clean-ups, it is general to use a much finer filament, and  $\frac{1}{750}$  (0.0013) in. is usual. The width of the shaving varies in practice from 0.04 to 0.1 in. The actual thickness and width should only be decided upon after due regard has been paid to working conditions, and the question of the method of cleaning-up taken into consideration.

M. A. Kapp<sup>1</sup> recommends the use of very fine flossy shavings in cases where copper and other base metals are present in such quantities as to interfere with the precipitation of gold and silver.

**Zinc-Box Practice.**<sup>2</sup>—Zinc-box capacity is calculated on the basis of the

<sup>1</sup> *Eng. and Min. Jour.*, Sept. 12, 1903.

<sup>2</sup> A. W. Allen, *Eng. and Min. Jour.*, Dec. 14, 1918.

number of tons of solution that can be satisfactorily precipitated per cubic foot of zinc, in position, per 24 hours. This datum is found empirically, but too much reliance should not be placed on the tests usually made to determine it. Zinc-boxes are cheaply constructed, and an additional two or three compartments in each box, over and above the number actually required, will be found an invaluable aid in simplifying operations and in shortening the time of clean-up.

Assuming ample capacity, a great saving of labour will be effected by adopting a system which takes into consideration the fact that the precipitating efficiency of a compartment of zinc shavings gradually decreases from the time circulation begins. If no excess capacity is provided, the adjusted amount of zinc will suffice to satisfy precipitation requirements for so short a time that perpetual dressing is unavoidable. This is undesirable, and may be prevented by the exercise of a little forethought.

A seven-compartment box is one of a number of types in common use. Experimentation may indicate that complete and satisfactory precipitation is possible with four freshly packed compartments. Further empirical data should be obtained to demonstrate the period which will ordinarily elapse before the barren solution gains in metal content, on account of the disintegration of zinc, short-circuiting of the solution, and the occlusion of the zinc surface with deposited gold and silver. A little investigation may show that 48 hours will elapse before there is appreciable rise in the grade of effluent solution. The fifth compartment in the box may then be packed; and, if this action staves off a rise in metal content in the effluent for 24 hours, the sixth compartment can be packed during the next day. The work of filling a clean compartment with fresh zinc is a matter of only a few minutes; and, by adopting a suitable modification of the plan outlined, a considerable saving of labour may often be effected, and the liability of mechanical loss of precious metal may be correspondingly reduced.

An objection to the aforementioned plan—that it will involve an undesirable distribution of precious metal over the surface of a large amount of zinc—does not hold good for about half the usual time between clean-ups, that is, during that period which immediately follows one of them. With the approach of a clean-up, it becomes necessary to dress the boxes more often, and so make conditions most favourable for the disposition of the maximum amount of precious metal with the minimum amount of zinc. Therefore, fresh zinc should never be placed in the first compartment. If a layer of coarser material is needed to ensure even percolation of the solution, or to form a bed for the short zinc which has accumulated, long filaments from one of the lower compartments, which are already plated, should be used.

The first compartment in a zinc-box need not always be filled with precipitant. It is highly advisable that the dissolving effect of the incoming solution, always greatest at its first contact with the shavings, should be

expended on short zinc, and this attacking action should be concentrated on the first compartment as much as possible before the clean-up, and the precious-metal sludge removed in a concentrated form. It is, therefore, better to reduce the depth of fine material in the first compartment, rather than to add too much coarse shavings.

Previous to the clean-up it is advisable to operate the boxes with a minimum of zinc consistent with satisfactory precipitation. Periods between dressings will become less as the date approaches. The final dressing before the clean-up should include the moving up of all the zinc in the lower compartments of the box, and the cleaning out of the last three or possibly four compartments, by transferring any cloudy solution, or sludge found below the screen, to the point of solution intake, so that the gold and silver precipitate in it can be recovered at the clean-up.

Just before the clean-up it is often practicable to fill two or even three of the lower compartments with fresh zinc. These will accumulate no appreciable quantity of metal during the space of 12 hours or so. When the busy day comes, not only will it be unnecessary to touch these compartments, but it will be possible to begin the flow of solution as soon as the upper compartments have been cleaned out, and the excess zinc has been returned.

In the majority of plants no extra skilled help is available at clean-up time, and attention must often be divided between the clean-up and regular routine duties. Under these conditions any plan that ensures labour-saving, without in any way impairing the results, should be encouraged by the superintendent. If the work connected with the monthly or semi-monthly clean-up can be distributed more evenly over the day preceding and that following, as well as over the busy day itself, operations will be found to proceed much more smoothly. The adoption of the suggestions outlined in the preceding paragraph resulted in a saving of 12 hours in cleaning-up operations in one plant, with the very reverse to impairment of efficiency. The zinc-boxes were all in operation before noon on the opening day of the clean-up, whereas previous practice had invariably resulted in delays that postponed transference of unpressed precipitate, the teasing of zinc, and the packing of the boxes far into the night. When important work is afoot, attention should be paid, in preliminary planning, to that arrangement of routine which eliminates the unnecessary, so that the chaos that too often accompanies a clean-up may be avoided.

The adoption of a logical system will obviate a great amount of labour, and save time and temper. It may be assumed that the contents of the first two compartments are fairly well down when the work begins. Immediately after the solution (or cleansing water) has been shut off, the amount of liquor in these two compartments should be reduced to a minimum, so that only sufficient remains to make the washing of any long zinc effective. The liquid may be lowered by bucketing the surplus over the edge of the first compartment, so that it passes into the bottom of the second, and

so on. Any coarse zinc remaining in the first compartment is then washed, before the screen is raised, and transferred to the second compartment. As soon as practicable, the valve or plug on the outlet pipe is opened, and the contents of the compartment are flushed to the clean-up sump, over which the necessary sieve is placed.

The screen used to support the zinc is replaced, after having been washed with a spray; and repacking is begun. A layer of plated zinc, taken from a lower compartment, forms a suitable bed, after which the coarse zinc is replaced, and the compartment finally filled with washed zinc from the second compartment, which is then cleaned out and filled in the same way. In lowering the solution level in subsequent compartments, any slightly cloudy liquid encountered can be returned to the first compartment.

The cleaning up of a number of units is merely a question of duplicating the method described. It is entirely unnecessary to permit trays and tubs of oxidising, steaming zinc to remain exposed in the manner frequently to be observed. Except as to the treatment of the material in the first compartment, already detailed, all zinc, as soon as washed, should be immediately packed in another compartment.

If the practice of cleaning out and filling up the final compartments just before the clean-up is adopted, it will, unless a considerable amount of zinc has to be removed, obviate the necessity for the handling of new zinc; or, in any event, only a small quantity will be needed on the day of the clean up. One or two empty compartments may be left in the centre of the box, but this is of no consequence, and the regular arrangement can be resumed as soon as the clean-up is over, and there is opportunity to attend to unimportant and temporarily deferred details.

**Rate of Flow.**—No definite rule can be laid down which might be available for the purpose of estimating the correct rate of flow of solution through a zinc-box. Such a figure would depend upon:

- (1) The degree of precipitation required;
- (2) The amount of metal to be precipitated;
- (3) The amount of free cyanide present in the solution; and
- (4) The condition of the solution with regard to zinc compounds and other impurities.

The actual amount may vary from 1 to 10 tons of solution per cubic foot of zinc per 24 hours; and data obtained from the results of actual practice with the particular solution will supply the only trustworthy information on the subject. The best rate having been determined, it is important that it should be kept as uniform as possible.

**Condition of the Solution.**—In order to ensure efficient precipitation of the contained metals, the solution passing through the boxes must be absolutely clear and entirely free from solids in suspension. Efficient filtration is a necessary preliminary to satisfactory working in the precipitation department.

From the chemical point of view it is necessary that the solution should contain free cyanide in amount sufficient to ensure satisfactory precipitation.

**Appearance of Zinc-Boxes.**—With a little practical experience the operator gets a rough general idea as to whether precipitation is effective or not. If the precipitate is loose, of a black or dark-brownish colour, it may be taken for granted that gold is precipitating satisfactorily. If, on the other hand, the deposit is firm and of lightish colour, it is possible that the sump solutions will assay high in gold.

In the cyanidation of silver ores the same appearances are available as indications of efficient precipitation, but in addition to a loose, black deposit there often occurs a grey deposit of almost pure silver. The quantity of the latter is usually dependent on the richness of the solution being precipitated.

If the appearance of the boxes indicates poor precipitation, the cyanide strength may be cautiously raised and results noted. The cumulative ill-effects of adding cyanide in lumps at the head of the zinc-boxes in an effort to improve precipitation is indicated by the subsequent determination for zinc content in the outgoing solution, and such practice should be avoided.

The white precipitate sometimes found in zinc-boxes is generally a mixture of ferro-cyanide of zinc and potassium, zinc cyanide, and zinc oxide. Its formation can generally be prevented by increasing the cyanide strength of the solution entering the zinc-boxes. In some cases such increase may be inadmissible, and there are numerous instances where the formation of the precipitate does not seriously interfere with precipitation and where little or no notice is taken of its presence in the zinc boxes. The presence of the precipitate is an evidence of one of the many complicated reactions occurring. A white precipitate of  $\text{CaO}$  also forms on the zinc, but only in the comparatively rare instance when lime has been added in considerable excess.

The usual white precipitate is more often found associated with the treatment of a pyritic ore and is one of the results of partial decomposition of the pyrites. In this connection it has been noticed that in the case of a highly pyritic ore, where the treatment included leaching of the sandy product and filter-pressing of the finer product, the white precipitate occurred only in those extractors dealing with the solution from the leaching plant. There was not a trace of white precipitate in the zinc-boxes of the slime plant, although the finer material carried an equal percentage of pyrites as the sand. This was doubtless due to the fact that little or no exposure to atmospheric oxygen took place in the slime plant, the pulp being agitated by mechanical means.

At the Tonopah Mining Company's mill at Nevada, a troublesome deposit on the zinc was found to contain ferrocyanide of zinc and calcium, brown hydrated oxide of manganese, and aluminium hydroxide. The only effective remedy was to clean-up frequently.<sup>1</sup>

**Efficient Precipitation.**—The precipitation of gold and silver from cyanide solutions serves two purposes. It recovers the metal sought for, and it

<sup>1</sup> E. J. Rice, *Eng. and Min. Jour.*, Jan. 17, 1911.

should leave a solution suitable as a wash for displacement purposes in the important operations of filtration or decantation.

Wash solution should carry as little metal of value as possible; and where the whole of the outgoing solution is used for washing purposes every effort should be made to effect almost complete precipitation of the metals during each passage through the boxes.

**Causes of Bad Precipitation.**—Inefficient precipitation may be due to insufficient zinc area or to too high a rate of flow through the boxes. Both of these causes are easily ascertainable and the remedies are obvious. When no improvement follows a larger exposure of zinc or a decreased rate of flow through the boxes, an analysis of the solution will probably reveal the cause of the trouble. When the commencement of operations has been marked, as it generally is, with good precipitation results, and when a steady reduction in percentage precipitation follows, the cause is, in all probability, due to a slow fouling of the solution. In the majority of instances there is, in ordinary working, an automatic limitation of zinc content as a result of various chemical combinations. Such reactions result in the ultimate disposal of an insoluble zinc compound with the residue. In other instances, however, the zinc content rises above an economical limit. In the latter event an alteration in the rate of flow through the boxes will generally cause no material improvement in the efficiency of precipitation, and a high metal content may be found in the solution after passage through the boxes. A temporary remedy for this condition may be found in the addition of more free cyanide to the solution entering the boxes, but the ill-effects of such a procedure are generally cumulative, and result in a further fouling of the solution and an increased consumption of zinc and cyanide. The only actual remedy lies in the absolute reduction in zinc content in the solution, and this may be ensured by the substitution of another precipitant, which forms no undesirable compounds with cyanide solution, in place of zinc, or by the adoption of electrical precipitation. Such a precipitant is aluminium.

TABLE XXXIX.—*Example of Progressive Rate of Precipitation.*

| Length of<br>Zinc Column<br>in feet. | Time of<br>Contact in<br>minutes. | Percentage<br>precipitated. | Length of<br>Zinc Column<br>in feet. | Time of<br>Contact in<br>minutes. | Percentage<br>precipitated. |
|--------------------------------------|-----------------------------------|-----------------------------|--------------------------------------|-----------------------------------|-----------------------------|
| 0.42                                 | 1                                 | 30.8                        | 5.42                                 | 13                                | 96.8                        |
| 0.83                                 | 2                                 | 51.9                        | 5.83                                 | 14                                | 97.1                        |
| 1.25                                 | 3                                 | 67.4                        | 6.25                                 | 15                                | 97.4                        |
| 1.67                                 | 4                                 | 78.1                        | 6.67                                 | 16                                | 97.7                        |
| 2.08                                 | 5                                 | 85.2                        | 7.08                                 | 17                                | 98.0                        |
| 2.5                                  | 6                                 | 89.7                        | 7.5                                  | 18                                | 98.3                        |
| 2.92                                 | 7                                 | 92.4                        | 7.92                                 | 19                                | 98.5                        |
| 3.33                                 | 8                                 | 94                          | 8.33                                 | 20                                | 98.7                        |
| 3.75                                 | 9                                 | 95.0                        | 8.75                                 | 21                                | 98.93                       |
| 4.17                                 | 10                                | 95.7                        | 9.17                                 | 22                                | 98.97                       |
| 4.58                                 | 11                                | 96.2                        | 9.58                                 | 23                                | 99.02                       |
| 5.0                                  | 12                                | 96.5                        | 10                                   | 24                                | 99.06                       |



In cases where the substitution of aluminium for zinc is not feasible, other methods must be adopted. Various suggestions involving electrical or chemical reactions have been brought forward. These include the electrolysis of the solution, or its treatment with free acid, carbonates, or potassium permanganate. Some of these suggestions may be available in isolated cases, but no continued success has been recorded as a result of the adoption of any of the above schemes. A practical method is to discharge periodically a small amount of plant solution and so keep the zinc content within workable limits. Such solution usually forms part or whole of the moisture percentage in the residue.

**Zinc-Box Data and Practice.**—In Table XL a number of figures dealing with zinc-box practice at various plants are given, and have been arranged in chronological order with reference to the dates of publication.

At the present time the tendency is to increase the rate of flow through the box and to diminish the volume of the shavings, paying greater attention to the packing and dressing of the boxes and the chemical and physical condition of the solution.

The question of zinc-box capacity has already been dealt with under 'rate of flow.' In the absence of definite information as to content of solution, no absolute rule can be laid down which might safely be adopted for the purpose of zinc-box design for any particular tonnage of solution. Five tons of solution per cubic foot of zinc per 24 hours may be taken as the average of a large number of installations.

The question has frequently been discussed as to which is preferable: a short column of large sectional area or a long column of small sectional area. If the cross-section is large and the column short, it is clear that either the velocity must be low, which is inadvisable, or the time of contact will be short. But the last column of Table XL shows that a long contact is not usually necessary, the total time of contact varying from 6 to 63 minutes. It is, however, well known that when boxes and solution are in good order, the first compartment extracts from 60 to 90 per cent. of the total, i.e. the greater part of the precipitation is effected in about 5 minutes in most cases. It follows, therefore, that a short column and a high velocity are not inadmissible, and that good work can be done under wide variation in the design of the boxes. With ordinary packing of the zinc, good results should be obtained under normal conditions with a column about 9 ft. long and a velocity of 0.5 ft. per minute. This would correspond to a sectional area of 0.044 sq. ft. for each ton treated daily. With close packing in hanks the velocity may be at least 1.5 ft. per minute and the corresponding cross-section 0.015 sq. ft. per ton per 24 hours.

In the first case the total flow will be 2.5 tons, and in the second case 7.5 tons of solution per day for each cubic foot of shavings.

**Consumption of Zinc.**—This may be considered under two heads:

(a) Mechanical losses, including the waste in cutting, also that destroyed in acid treatment and in smelting.

TABLE XL.—*Variations in zinc-box Practice.*

|                               | Cyanide Strength. | Assay Contents before Precipitation. |        | Assay Contents after Precipitation. |        | Cubic feet of Zinc per ton of Solution precipitated per 24 hours. | Solution per 24 hours for each cubic foot of Zinc. | Cross-section of Zinc Column per ton of Solution in 24 hours. | Distance traversed through Zinc. | Nominal Velocity of Flow, i.e. cubic feet per minute ÷ cross-section. | Time of Contact with Zinc. | Authority.  |
|-------------------------------|-------------------|--------------------------------------|--------|-------------------------------------|--------|---|--|---|----------------------------------|---|----------------------------|---|
|                               | Per cent.         | Au dwt.                              | Ag oz. | Au dwt.                             | Ag oz. | Cubic feet.   | Short tons.  | Sq. feet.   | Lineal feet.                     | Feet per minute.  | Minutes.                   |   |
| Rand, 1903, . . . . .         | ...               | 0.75                                 | ...    | 0.06                                | ...    | 1.01  | 0.99   | 0.0674  | 15                               | 0.33  | 45                         | A. P. Griffith and F. W. Oldfield.<br>E. M. Hamilton. |
| Mexico, 1903, . . . . .       | 0.30              | 0.80                                 | 2.00   | tr.                                 | 0.05   | 0.98  | 1.07   | 0.0714  | 14                               | 0.38  | 42                         |   |
| " 1906, . . . . .             | 0.10              | 0.97                                 | 3.41   | tr.                                 | tr.    | 0.14  | 7.27   | 0.0122  | 11                               | 1.33  | 6                          |   |
| " 1908, . . . . .             | 0.15              | ...                                  | ...    | ...                                 | ...    | 1.40  | 0.71   | 0.16  | 8.8                              | 0.14  | 63                         |   |
| Barberton, Transvaal, 1909, . | ...               | 3.18                                 | ...    | 0.01                                | ...    | 0.38  | 2.63   | 0.0873  | 4.4                              | 0.25  | 18                         | Claude T. Rice.                                       |
| Knight's Deep Rand, 1910, .   | 0.03              | 2.56                                 | ...    | 0.01                                | ...    | 0.65  | 1.54   | ...   | ...                              | ...   | ...                        | Mather Smith.   |
| Tonopah, Belmont, 1911, .     | 0.25              | 0.68                                 | 6.36   | 0.03                                | 0.30   | 0.42  | 2.40   | 0.042   | 10                               | 0.53  | 19                         | F. D. Phillips.<br>Claud T. Rice.                     |

(b) Chemical losses occurring in the extractor boxes.

The actual amount of zinc dissolved during precipitation is a matter that is not easy to determine, but it is certainly many times in excess of the theoretical quantity required. If the whole of the zinc dissolved be used to precipitate the gold or silver, then 1 oz. should deposit 6 oz. of gold or 3.3 oz. of silver. This proportion would apply in the case of a concentrated solution of  $\text{KAu}(\text{CN})_2$  or  $\text{KAg}(\text{CN})_2$ , for then the molecules of the salt are so numerous that as soon as one is decomposed and the metal precipitated, fresh molecules instantly come into contact with the metallic surface and are also decomposed as the zinc dissolves. In the cyanide process, however, the solutions in use generally contain about 1 part of gold in 100,000 to 1,000,000 parts of solution, so that the molecules of the salt are so far apart as to make it impracticable to bring the gold-bearing molecules in contact with the metallic surface as fast as required by the dissolving zinc. The result is that there are a hundred or more zinc molecules for every molecule of the gold salt that strikes this surface at the right spot and at the right time. The zinc is, therefore, for the most part wasted in decomposing water and liberating hydrogen.

A number of circumstances affect the consumption of zinc in a cyanide plant. In the cyanidation of silver ores the amount of metal to be precipitated has an important bearing on the consumption of zinc. Another point influencing the consumption is the amount of solution being handled by the extractors per ton of ore treated. Thus in a milling-in-water plant the amount of solution to be precipitated may amount to two or three times the weight of ore treated. In a milling-in-cyanide plant the amount of solution to be precipitated must, by the return-flow process, approach the milling dilution which, in many instances, may reach a much greater amount.

The condition of the solution with regard to cyanide and alkalinity strength is also an important factor in zinc consumption. An unnecessarily high cyanide and free alkali strength means an abnormal zinc consumption. This fact is evidenced in several instances in the cyanidation of silver ores where the zinc consumption, on account of high cyanide strength, is altogether out of proportion to the amount of metal deposited.

In actual practice the consumption of zinc varies from 0.10 lb. per ton treated to 3 or even 4 lb. In the former case the treatment may comprise the handling of a very low-grade slime from gold-ore treatment. In the latter case the treatment may comprise the milling-in-cyanide of a high-grade silver ore.

In Table XLI. are given some figures of zinc consumption with the calculated approximate amount consumed chemically and mechanically in each case.

**Zinc-Lead Couple in Practice.**—The physical condition of the negative surface has already been referred to as one of the factors affecting precipitation. In strong cyanide solutions, or in solutions containing a large percentage of other salts, this factor is not very apparent, for the reason

that the gold deposits in a manner tending to produce a negative surface of the requisite physical condition. On the other hand, and when the solution is very weak in salts, the gold deposits as a hard film which raises the decomposition point, as already described.

TABLE XLI.—*Zinc Consumption.*

| Material treated. | Dwt. of Ag or Au in Solution. | Lb. of Zinc per ton of Material treated. |             |        | Lb. of Zinc per ton of Solution. |             |        | Lb. of Zinc per oz. of fine Bullion. |             |        | Calculated from Data published by |
|-------------------|-------------------------------|--|-------------|--------|----------------------------------|-------------|--------|--------------------------------------|-------------|--------|-----------------------------------|
|                   |                               | Chemical.                                | Mechanical. | Total. | Chemical.                        | Mechanical. | Total. | Chemical.                            | Mechanical. | Total. |                                   |
| Sand, .           | ...                           | 0.17                                     | 0.05        | 0.22   | ...                              | ...         | ...    | 1.15                                 | 0.35        | 1.50   | M. Smith, 1909.                   |
| Slime, .          | 2.71                          | 0.39                                     | 0.67        | 1.06   | 0.07                             | 0.11        | 0.18   | 0.52                                 | 0.81        | 1.33   | W. H. Virgoe, 1909.               |
| Sand, .           | 11.25                         | 0.91                                     | 0.15        | 1.06   | 0.52                             | 0.08        | 0.60   | 0.92                                 | 0.15        | 1.07   | " "                               |
| Roasted telluride | 20.00                         | ...                                      | ...         | ...    | 0.33                             | 0.59        | 0.92   | 0.33                                 | 0.59        | 0.92   | P. Argall, 1895.                  |

*Notes.*—The theoretical amount of zinc required to replace 1 oz. of gold is 0.011 lb., and 1 oz. of silver, 0.021 lb.

W. K. Betty observed that when a zinc-lead couple was used with very weak cyanide solutions, even in the absence of an appreciable quantity of other salts, as is the case in the treatment of slime, a satisfactory precipitation could be obtained. The use of the couple had been previously patented by MacArthur, but when the invention was applied to the strong solution then used in sand treatment, its value was not appreciated because the gold deposited in a loose powdery form, and therefore the zinc-lead couple was unnecessary.

The method of preparing the zinc-lead couple is by immersing the ordinary zinc shavings in a solution of lead acetate, containing from 1 to 5 per cent. of lead, just before being placed in the precipitation box. By this treatment, metallic lead is precipitated in a loose spongy form on the zinc to the extent of about 1 per cent. of the weight of the latter.

**Zinc-dust Precipitation.**—This method was introduced by H. L. Sulman in 1894. In its first application the solution was separated from the precipitate by decantation and settlement. The next step in advance was the use of a filter-press to collect the precipitate; but the successful application of the process is due to C. W. Merrill, who has systematised the method and holds controlling patents for improvements in application.

The Merrill process consists of the emulsification of the gold- and silver-bearing solutions with zinc fume, with special provision for the exclusion of air after the contact of the fume with the solution. Intimate mixture of solution and precipitant is effected by the pumping system which delivers

into a specially designed press. Additional agitation can be arranged for by the by-passing of the mixture so that additional circulation can be ensured. The precipitation press has frames of triangular shape which discharge the solution at their top level, so that there is no oxidation of the unconsumed zinc in the press. The precipitated solution passes through the cloth and returns to storage tanks, whence it again follows the usual course throughout the plant.

In cases where it is not necessary to precipitate the whole of the solution to barrenness, two separate circuits are employed. In one of these a percentage of the solution is treated with the usual excess of zinc dust, and a low-grade residue solution ensured. This barren solution is necessary for wash purposes in filtration.

In the second circuit the solution is treated with less zinc than would be required to effect complete or almost complete precipitation. This solution is deprived of the bulk of its gold and silver content and is returned to the main circuit. The precipitate from the incomplete precipitation circuit is of higher grade than that obtainable from the complete precipitation circuit.

At the Trojan plant, Black Hills,<sup>1</sup> this method was used to separate the copper in solution from the bulk of the gold. A preliminary precipitation was made with dust, and the solution from the press was then run through shavings. The press product yielded 80 per cent. of bullion, 936 fine, and the greater part of the copper passed to the zinc-box.

The zinc dust ordinarily used is a by-product of the distillation process for the extraction of zinc from its ores. The dust is sieved through an 80-mesh screen to eliminate coarse particles of zinc, and the marketable product contains about 90 per cent. of metallic zinc, 8 per cent. of zinc oxide, and some lead, cadmium, arsenic, and antimony.<sup>2</sup>

Burr A. Robinson<sup>3</sup> mentions a patented process for producing atomised zinc in a comparatively small apparatus, 2 ft. diameter by 4 ft. high, with a collecting chamber 3 ft. diameter by 18 ft. long, suitable for installation in cyanide plants.

**The Deoxidisation of Cyanide Solutions prior to Zinc Precipitation.**—In the annual review number of the *Engineering and Mining Journal*, January 11, 1919, attention is drawn<sup>4</sup> to a comparatively recent modification in connection with the precipitation of gold and silver which received publicity during 1918. A brief account of T. B. Crowe's experiments and the practical results achieved by deoxidising cyanide solutions by means of a vacuum, prior to zinc-dust precipitation, was published,<sup>5</sup> but few details were given. It is to be noted, however, that considerable savings have been effected in the consumption of zinc dust and cyanide.

<sup>1</sup> Jesse Simmons, *Eng. and Min. Jour.*, Aug. 19, 1911.

<sup>2</sup> Paul Speier, *Breslau Min. Jour.*, Nov. 26, 1910.

<sup>3</sup> *Eng. and Min. Jour.*, Jan. 6, 1912.

<sup>4</sup> *Metallurgy of Gold and Silver*, by A. W. Allen.

<sup>5</sup> *Trans. A.I.M.E.*, 1918.

Mr Crowe's explanation of the theory of the reactions involved assumes that nascent hydrogen is the active precipitating agent, and that its combination with oxygen in the solution reduces the precipitating efficiency. This view is opposed by O. P. Watts,<sup>1</sup> who draws attention to the fact that the dissolving of zinc in a solution of pure sodium cyanide is an example of the displacement of one element by another; and that, when zinc is immersed in a dilute cyanide solution containing gold, two reactions occur simultaneously—some of the zinc is dissolved by the double cyanide of gold with precipitation of the precious metal, and a portion of the zinc dissolves in the sodium cyanide, forming hydrogen.

The distinctive feature of the new modification is undoubtedly the saving of zinc and cyanide resulting from the feasibility of abolishing the practice of adding lump cyanide at the head of the presses. In this connection it is interesting to note that the Crowe modification is a radical departure from the practice, so often found beneficial in improving precipitating efficiency with zinc-shaving precipitation, of *oxidising* the solution, before it reached the boxes, by treatment with air. The purpose was doubtless to accelerate action by ensuring the prompt removal of occluded hydrogen from the surface of the zinc shavings, and so exposing fresh areas of the precipitant. Such a method would be unlikely to ensure any economy in the consumption of either zinc or cyanide, although improved precipitation results were often obvious.

The theory that the presence of oxygen assists in depolarisation is questioned by H. A. White, who has contributed<sup>2</sup> a new method for the estimation of this element in cyanide solutions. Typical analyses of solution in a Rand zinc-box showed that there was little precipitation of gold in the first compartment, although the percentage of oxygen in the solution was high. In this instance, however, other factors are to be taken into consideration. The first compartment of the box may contain the bulk of the white precipitate common in that district, so that vigorous action would be unlikely in any case. The white precipitate is found when free cyanide content is low, and it would be interesting to know whether its formation could be prevented by means of a preliminary deoxidising of the solution.

The whole subject of the reactions involved during the precipitation of gold in aurocyanide solution by zinc shavings or dust constitutes a field for investigation, particularly in relation to the interfering or helpful influence of oxygen. Comparative figures will doubtless be available at a later date, and a more complete hypothesis may be suggested to account for the results.

**Copper and Poor Precipitation.**—Much difficulty has often been caused by the presence of copper in the solution. Zinc in a cyanide solution precipitates copper, but the potential difference between the two metals is

<sup>1</sup> *Chem. and Met. Eng.*, Nov. 1, 1918, p. 652.

<sup>2</sup> "The Estimation of Oxygen in Working Cyanide Solutions," *Jour. Chem. Met. and Min. Soc. of S. Africa*, June 1918, p. 292.

small compared with that between zinc and gold, or zinc and silver. The effect of this is that a solution which would just precipitate a loose black deposit of gold would not be at all suitable for producing a loose deposit of copper. If copper were present in such a solution it would be precipitated as a firm metallic coating, such as would raise the decomposition point of the solution above that which the potential of the zinc was capable of effecting, and then the action would fall off. Should the solution be sufficiently strong in salts to deposit the copper in a loose form, both gold and copper precipitate together; but as the solution pressure of copper is higher than that of gold, a large proportion of the former is redissolved. Nevertheless the zinc-gold couple is, to a large extent, replaced by a copper-gold couple which is less effective in precipitating both gold and silver, and the net result is a poorer precipitation.

The strength of the solution required to produce the loose deposit is rarely met with except in silver cyanidation. In most cases a moderate increase in cyanide strength from the normal only tends to keep the copper in solution.

A paper by Robert Lindsay<sup>1</sup> describes the practice at a Transvaal gold mine where it was found advisable to use weak cyanide solutions (0.04 per cent.) and lengthy leaching treatment. According to this account the cyanide consumption is seldom under 0.8 lb. per ton and sometimes as high as 2.5 lb., according to the copper content of the ore. A preliminary wash with weak sulphuric acid to remove the copper is prohibitive in cost, the consumption of acid being about 30 lb. per ton, owing to the presence of dolomite and iron oxide. The average recovery for the last twelve months was 83 per cent.

The slime is treated by a combination of the decantation and Usher processes. Instead of using sand filters as clarifiers for the solution from the slime plant, the solution is allowed to flow quietly into two settlers (18 by 5 ft.), the first having one and the second two partitions of coco matting suspended from iron bars laid across the vats transversely to the flow. The matting is fastened to the sides and rests on the bottom. These settlers deal with 1000 tons of solution per day for sixteen months before requiring cleaning out. They are then cleaned out with a fire hose and are ready again for use in 3 or 4 hours. The average recovery from slime is 87.5 per cent.

Cyanide solutions containing copper, deposit this metal readily on clean zinc; the copper forms a closely adherent metallic film or plating, which after a time coats the zinc so completely that the precipitation of gold almost ceases. The lead-zinc couple is used at Pilgrim's Rest with advantage for cupriferous solutions. Lead-coated zinc does not precipitate copper so readily from working solutions as clean zinc, and the object aimed at is not to precipitate all the copper, but to keep the amount of copper in solution more

<sup>1</sup> "Notes on the Treatment of Pilgrim's Rest Ore," *Jour. Chem. Met. and Min. Soc. of S. Africa*, April 1917.

or less constant. If the lead coating be too light, too much copper will be thrown down, and more zinc will have to be added to the extractors before cleaning up, resulting in an unnecessary consumption of zinc. The practice in dipping is to regulate the quantity of zinc dipped with the strength of the lead solution so that each 100 lb. of zinc carries 5 lb. metallic lead. That suits the copper content of the solution and precipitates about one-half of the copper entering the extractors, whereas the gold content of the effluent is low even in the cold season. Increasing the cyanide strength as a remedy for bad precipitation is not recommended. It gives temporary relief, no doubt, by keeping copper in solution, but the gradual accumulation affects extraction, and the excess copper has to be eliminated eventually, either by precipitation or by running some of the solution to waste.

If the soluble copper content of the ore should become abnormally high, it will be found advantageous to have uncoated zinc in the first compartment to precipitate a proportionally large amount of copper, leaving the remaining compartments of lead-coated zinc to precipitate the gold and a portion of the remaining copper. The first compartment of coppery zinc is taken out every four or five days and immersed in dilute acid for a few minutes, and returned to the box together with enough zinc to fill the compartment. There is an increase in zinc consumption, but the gold precipitation is good.

The precipitation of copper on zinc is, of course, accompanied by a regeneration of cyanide, thus :



or



consequently the cyanide strength leaving the extractor box is always higher than that entering it, as shown in the table.

The tendency of the copper to remain in the stronger solution of the sand boxes will be noted, the precipitated copper there being 41·7 per cent. as against 44·4 per cent. in the case of the weaker cyanide solution of the slime boxes with a lower copper content entering. One cubic foot of zinc is allowed per ton of solution per 24 hours.

TABLE XLIA.—*Strength of Solution entering and leaving Zinc-Boxes.*

|                         | Sand Boxes.       |                   |                     | Slime Boxes.      |                   |                     |
|-------------------------|-------------------|-------------------|---------------------|-------------------|-------------------|---------------------|
|                         | Per cent.<br>KCN. | Per cent.<br>CaO. | Cu, lb.<br>per ton. | Per cent.<br>KCN. | Per cent.<br>CaO. | Cu, lb.<br>per ton. |
| Entering . . . . .      | ·032              | ·007              | ·12                 | ·022              | ·008              | ·09                 |
| After 1st Comp. . . . . | ·039              | ·008              | ·11                 | ·025              | ·011              | ·07                 |
| „ 2nd „ . . . . .       | ·044              | ·009              | ·08                 | ·028              | ·010              | ·05                 |
| „ 3rd „ . . . . .       | ·042              | ·010              | ...                 | ·029              | ·010              | ...                 |
| „ 4th „ . . . . .       | ·040              | ·010              | ...                 | ·028              | ·011              | ...                 |
| „ 5th „ . . . . .       | ·039              | ·011              | ·07                 | ·028              | ·011              | ·04                 |



The clean-up takes place twice a month. The top compartment only of each box is taken out and treated as usual with sulphuric acid. When all the zinc has been dissolved, the copper is attacked by nitric acid, or with sulphuric and nitric acids together.

Over 700 lb. of copper is dissolved out every clean-up. The acids are added in the proportion of 1 part sulphuric to 0.44 part nitric acid, and the temperature should not be under 60° C. if copper is to be kept out of the bullion. The precipitate, kept in suspension by mechanical stirrers, is tested for the presence of copper after each addition of acid, so as to ensure no more acid being added than is absolutely necessary. When the copper is all in solution, the vat is filled with water and a few pounds of size added to assist settlement, which usually takes place in 5 or 6 hours. The solution containing the zinc and copper is passed through a sand filter to catch any fine gold slime in suspension. The surface of the vat is skimmed every other month and the sand used as flux.

The coppery solution now flows through a stoneware pipe to two precipitating vats, which are filled with scrap iron. Any old iron is put into these—grate bars, fire doors, worn-out plates, or battery screening. Every four months the vats are cleaned out and the precipitate screened to keep out coarse iron. It is then air dried, thoroughly mixed, sampled, assayed, and bagged in sugar sacks. A shipment of this precipitation assayed 10.99 oz. Au, 123.79 oz. Ag per ton, and 40.95 per cent. Cu.

There is no gold or silver in the copper-vat effluent, and only a trace of copper toward the end of the run. It is found that coarse iron gives a much purer product, and by using that exclusively it is hoped to bring the precipitate up to over 60 per cent. grade. The copper recovered is equal to 0.13 lb. per ton milled, and agrees fairly closely with the difference between the copper content of originals and residues.

The washed gold slime is filter-pressed and dried. Calcining is not found necessary after using nitric acid. The gold slime assays about 30 per cent. Au and is fluxed with 25 per cent. borax, 12½ per cent. sand, and 12½ per cent. MnO<sub>2</sub>. The bullion averages 850 fine. The average total recovery from the plant is 94 per cent.

## CHAPTER XXX.

### ELECTRICAL PRECIPITATION.

**General Considerations.**—Electro-chemical action takes place in all precipitation processes now in use, but the term 'electrical precipitation' refers only to those systems in which the current is supplied from an external source.

It has already been pointed out that, when a current is applied to a cyanide solution containing gold and silver, the K ions travel towards the cathode and the  $\text{Au}(\text{CN})_2$  or  $\text{Ag}(\text{CN})_2$  ions towards the anode. The necessity for keeping the solution in motion has also been demonstrated, so that the gold- and silver-bearing ions are brought into contact with the cathode. Some of these ions, however, do reach the anode, where they give up their negative charges and separate as atoms. Thus when iron plates are used as anodes some gold may be found associated with the ferric oxide formed.

The amount of current used, when electro-plating in cyanide solution, is roughly proportional to the quantity of gold per unit of solution. A good deposit is obtained with 10 amperes per sq. ft. in a solution containing 50 oz. of gold per ton. This is equivalent to 0.01 ampere per sq. ft. for each pennyweight per ton.

After ore treatment and with, say, a solution of 4 dwt. of gold per ton, a current of 0.05 ampere per sq. ft. will give an adherent deposit. This is equivalent to 0.013 ampere per sq. ft. for each pennyweight per ton of solution.

As the relation of current density to solution value is nearly the same throughout such a wide range, it may be assumed that such a ratio will hold good with solutions of still lower grade. For extremely low-grade gold solutions the current required to obtain an additional deposit would be less than when dealing with high-grade solutions of silver.

In the Siemens-Halske process an adherent deposit is necessary. In some other processes it is not essential.

### THE ELECTRODES.

**Anodes.**—These may be divided into three classes :

1. (a) Those soluble in the cyanide solution, or (b) in an electrolyte separated from the solution by a porous partition.
2. Those which are not easily soluble.
3. Those which are practically insoluble.

*Class 1.* The advantage of soluble over insoluble anodes is that a smaller external e.m.f. is required to produce the same result. A prominent dis-

advantage is the continuous consumption of anode. A further disadvantage lies in the difficulty found in regulating the e.m.f. This regulation is necessary in order to prevent the formation of non-conducting coatings on the anode and also the deposition of the anode metal, together with the gold and silver, on the cathode.

To obviate the above-mentioned disadvantages the use of a porous partition has been proposed, with the cyanide solution containing the cathode on the one side, and the anode in a suitable solvent on the other.

It should be noted, however, that no method of using soluble anodes has met with success.

**Class 2.**—Iron is the most important metal of the partly soluble class. In a strong cyanide solution and a weak current, the compound formed is chiefly  $K_4Fe(CN)_6$ . In a dilute solution and with currents employed in practice, the K and CN ions are insufficient to carry the whole current, with the result that the water, and other compounds which act as electrolytes, are decomposed. The oxygen thus liberated gradually produces a thick crust of hydrated oxides of iron on the anode, which afterwards scales off. Compounds of iron and cyanogen are produced to a smaller extent, Prussian blue being an example.

**Class 3.**—Gas carbon is practically insoluble, but in all cases where oxygen is liberated at the anode the carbon disintegrates and forms a black substance called mellogen, as well as CO and  $CO_2$ .

Graphite also disintegrates under similar conditions and becomes partly oxidised, but no mellogen is formed.

**Peroxide of Lead Anodes.**—These have been successfully used in practice. Lead plates, if properly peroxidised, have good crystalline surfaces, are only slowly acted on by cyanide solution, and last a long time. Badly prepared plates will soon blister even with a weak current (0.06 ampere per sq. in.) and will fall to pieces.

The use of lead peroxide possesses a distinct advantage over iron as anode material, in that no sludge is formed in the boxes; and, as no sackcloth covers are required, the anodes and cathodes can be placed closer together.

**Lead-Antimony Anodes.**—A lead-antimony alloy, containing 6 to 9 per cent. of antimony, has been used in Mexico with satisfactory results.

**Cathodes.**—Lead foil was successfully used in the Siemens-Halske process to be briefly described. Carbon in thin blocks would be effective but is fragile and costly.

Iron plates coated with plumbago or waxed conducting surface, so that the precipitated metal could be stripped off, were tried by Julian in 1894, but the method was only available with rich solutions. He also experimented with fibrous and textile materials, rendered conductive by carbonising or other method, but no practicable process resulted.

Zinc cathodes were tried by Andreoli without success. The same investigator used iron plates from which he removed the gold by dipping them into a bath of molten lead covered with oil.

Aluminium was proposed by S. Cowper-Coles.<sup>1</sup> Julian also experimented with plates of this material, but found that the difficulty of removing and collecting the thin deposits formed only one of several reasons against their adoption.

Suggestions involving the use of amalgamated copper plates have been brought forward so frequently that a few words of warning are necessary.

Sir T. K. Rose<sup>2</sup> has shown that with solutions carrying 10 oz. per ton and a current density exceeding 0.03 ampere per sq. ft. the gold was thrown down faster than it could amalgamate and was found lying on the surface of the mercury in the form of a black powder. This showed the necessity for a large cathode surface, even with rich solutions.

The difficulties met with in practice in connection with the use of amalgamated copper plates may be summarised thus :

(1) With a surface similar in extent to that used in the case of lead cathode, the cost of copper and mercury is excessive.

(2) A portion of the gold is permanently retained by the cathode.

(3) The plates become corroded and are in time destroyed. This is probably due to the joint action of the mercury, and the potassium or sodium deposited from the solution.

#### ELECTRICAL PRECIPITATION IN PRACTICE.

Years of experience have shown that electrical precipitation cannot economically compete with a zinc or equivalent process ; and the use of the former is only justified in those exceptional cases where other methods have proved unsatisfactory.

**The Siemens-Halske Process.**—In this excellent adaptation of electrical methods the anodes used were iron plates  $\frac{1}{8}$  to  $\frac{1}{4}$  in. thick. These were sewn up in hessian covers, to prevent short-circuiting by the falling of scales of oxide.

The cathodes were of lead foil 0.0032 in. thick and weighing 0.19 lb. per sq. ft. These were hung from horizontal iron wires over which the top edge of the sheet was turned, and sewn together with fine copper wire. The space between anode and cathode was usually about  $1\frac{1}{2}$  in.

For details see Tables XLII. and XLIII. Fig. 47 gives the percentage of gold electrically precipitated after equal intervals of time, as calculated from tests by J. R. Williams, and shows how rapidly the rate of precipitation diminishes as solution content is reduced.

The average anode surface of a number of installations amounted to 60 sq. ft. for each ton of solution treated per day ; and the net space occupied by the boxes was 17 cu. ft. per ton per 24 hours.

The consumption of iron averaged 0.5 lb., which included the waste in worn-out plates. The oxide sludge produced was about 0.2 lb. per ton of solution treated.

<sup>1</sup> *Trans. Inst. Min. and Met.*, vol. vi. p. 219.

<sup>2</sup> *Ibid.*, vol. viii. p. 369.

TABLE XLII.—Dimensions of Boxes and results obtained with the Siemens-Halske method of Electrical Precipitation.

|   |                  |                  |               |                               |                  |                       |                       |
|---|------------------|------------------|---------------|-------------------------------|------------------|-----------------------|-----------------------|
| Gold per ton after precipitation in grains.                                   | ...              | ...              | trace         | 18                            | 24               | 21                    | 8                     |
| Gold per ton before precipitation in grains.                                  | ...              | ...              | 48            | 120                           | 24               | 93                    | 17                    |
| Specific rate of flow, i.e. cubic feet per minute per square foot of section. | ...              | 0.041            | ...           | 0.059                         | 0.089            | 0.086                 | ...                   |
| Rate of flow in cubic feet per minute.  | ...              | 1.56             | ...           | 6.67                          | 7.11             | 8.42                  | ...                   |
| Length of column in feet.   | ...              | 28               | ...           | 24                            | 44               | 48                    | 48                    |
| Section of liquid column in square feet.                                      | ...              | 38               | ...           | 114                           | 80               | 40                    | 186                   |
| Cathode surface in square feet.   | 12672            | 5274             | 7680          | ...                           | ...              | ...                   | ...                   |
| Anode surface in square feet.   | 14784            | 5040             | 3240          | 13824                         | 16896            | 9216                  | 38024                 |
| Total cubic feet occupied by electrodes.                                      | 1512             | 770              | 1080          | 2736                          | 3520             | 1920                  | 8980                  |
| Cubic feet in boxes per ton of solution.                                      | 16.8             | 16               | 25            | 14.25                         | 18.13            | 20.13                 | ...                   |
| Total tons of solution in 24 hours.   | 120              | 70               | 50            | 800                           | 320              | 154                   | ...                   |
| Total cubic feet in boxes.  | 2016             | 1125             | 1249          | 4276                          | 5800             | 3100                  | 13960                 |
| Depth of each box, in feet and inches.  | 4' 0"            | 4' 0"            | 9' 0"         | 5' 0"                         | 5' 0"            | 5' 0"                 | 5' 0"                 |
| Width of each box, in feet and inches.  | 7' 0"            | 5' 2½"           | 4' 7½"        | 4' 9"                         | 10' 0"           | 5' 0"                 | 15' 0"                |
| Length of each box, in feet and inches.                                       | 18' 0"           | 18' 6"           | 30' 0"        | 30' 0"                        | 29' 0"           | 31' 0"                | 30' 0"                |
| Amps per square foot.   | 0.047            | 0.06             | ...           | 0.04                          | 0.04             | 0.04                  | 0.04                  |
| Number of boxes.  | 4                | 4                | 1             | 6                             | 4                | 4                     | 3                     |
| Per cent. KCN in solutions.   | .15 to .05       | .2 to .02        | .15 to .05    | .08 to .02                    | .08              | .10 to .03            | .017                  |
| Name of Works.  | Worcester, . . . | Gibraltar, . . . | Cresna, . . . | May Consolidated, sand, . . . | " " slime, . . . | Lancaster West, . . . | Pereira, slime, . . . |
|   |                  |                  |               |                               |                  |                       | " "                   |

TABLE XLIII.—Showing area of either Electrode required in the Electrical Precipitation of 1 ton of solution in 24 hours, when working with anodes and cathodes of equal surface.

| Original Content of Solution. Grains per ton. | Gold Content of Solution after Precipitation, in grains per ton. |      |      |      |       |       |               | Trace. |
|---|--|------|------|------|-------|-------|---------------|--------|
|   | 20   | 16   | 12   | 8    | 4     | 2     | $\frac{1}{2}$ |        |
| 84  | 48   | 58   | 70   | 90   | 112   | 138   | 154           | 176    |
| 80  | 47   | 57.5 | 69   | 89   | 111   | 132   | 153           | 174    |
| 78  | 46.5   | 57.3 | 68.5 | 88.5 | 110.5 | 131.5 | 152.5         | 173.5  |
| 76  | 46   | 57   | 68   | 88   | 110   | 131   | 152           | 173    |
| 72  | 45   | 56   | 67   | 87   | 109   | 130   | 151           | 172    |
| 68  | 44   | 55   | 66   | 86   | 108   | 129   | 150           | 171    |
| 66  | 43.5   | 54   | 65   | 85.5 | 107.5 | 128.5 | 149.5         | 170    |
| 64  | 43   | 53   | 64   | 85   | 107   | 128   | 149           | 169    |
| 60  | 42   | 52   | 63   | 84   | 105   | 126   | 147.5         | 168    |
| 56  | 40   | 51   | 62   | 83   | 103   | 124   | 146           | 167    |
| 54  | 39.5   | 50.5 | 61   | 82   | 102.5 | 123.5 | 145           | 166    |
| 52  | 39   | 50   | 60   | 81   | 102   | 123   | 144           | 165    |
| 48  | 37   | 48   | 58.5 | 79   | 100   | 121   | 142           | 163    |
| 44  | 34   | 45   | 57   | 77   | 97    | 118   | 139           | 160    |
| 42  | 33   | 44   | 55   | 76   | 95.5  | 117   | 137.5         | 158.5  |
| 40  | 32   | 42   | 53   | 74.5 | 94    | 116   | 136           | 157    |
| 36  | 29   | 39   | 50   | 71   | 91    | 112   | 132           | 153    |
| 32  | 25   | 35.5 | 46   | 67   | 87    | 108   | 128           | 149    |
| 28  | 21   | 31.5 | 42   | 62   | 82    | 103   | 123           | 144    |
| 24  | 15   | 25   | 35   | 56   | 76    | 96    | 116           | 137    |
| 20  | ...  | 16   | 27   | 48   | 66    | 87    | 107           | 128    |
| 16  | ...  | ...  | ...  | 34   | 54    | 76    | 96            | 117    |
| 12  | ...  | ...  | ...  | ...  | 40    | 61    | 82            | 108    |
| 8   | ...  | ...  | ...  | ...  | 22    | 43    | 64            | 86     |
| 4   | ...  | ...  | ...  | ...  | ...   | ...   | 43            | 66     |
| 2   | ...  | ...  | ...  | ...  | ...   | ...   | ...           | 43     |

Note.—Find horizontal line corresponding to known original content and vertical column under the residual value. At the intersection of these two is the area required.

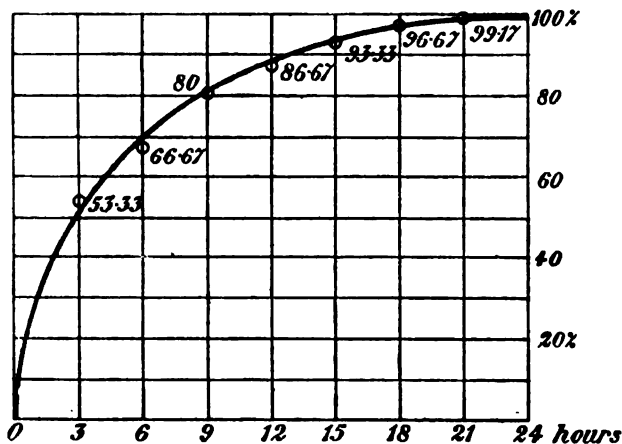


FIG. 47.—Rate of Precipitation of Gold.



to 0.0146 per cent. during the passage of the solution through the box. This amounts to over  $\frac{1}{4}$  lb. per ton of solution treated. The solution leaving the boxes contained 6 oz. of copper per ton.

The Butters process was also introduced at the San Sebastian plant, where a large amount of copper was present. The compact deposit of gold and copper formed was difficult to scrape off the tinned surface of the cathode. Lead-foil cathodes were tried, but refining by cupellation was impracticable owing to the large excess of copper. The difficulties were subsequently solved by C. P. Richmond,<sup>1</sup> the metallurgist in charge, in the following manner:

The cathodes, of  $\frac{1}{16}$ -in. sheet lead, were hung in the box at about  $2\frac{1}{2}$ -in. centres, so that the space between anode and cathode was about  $1\frac{1}{8}$  in., requiring an e.m.f. of from 4 to 4.5 volts.

Under these conditions, the deposit produced is hard and adherent, and the sheets can be readily handled without loss. A small quantity of low-grade precipitate collects on the anodes and also in the bottom of the box, and carries from 5 to 50 oz. per ton.

After 30 days the precipitate amounted to 12 lb. The cathodes were then removed, drained, and placed as anodes at 8-in. centres in a refining box containing a 2 or 3 per cent. sulphuric-acid solution. Each plate was hung in a wooden frame, over which was stretched a cotton-cloth sack.

The cathodes in this box were also of  $\frac{1}{16}$ -in. sheet lead. With 8 volts the current density was 5 amperes per square foot. The copper on the plate dissolves, passes through the cloth, and is thus separated from the gold which falls to the bottom of the wooden frame, within the sack. The plates were taken out when clean, drained, and placed on a washing board over the clean-up tank. The cloth sack was cut down, washed, and removed. The gold slime was washed into the tank and afterwards drained, dried, and smelted. The plates were then ready for use again as cathodes in the precipitating box.

The copper slime was flushed out through the  $1\frac{1}{2}$ -in. holes in the refining box, collected in a small filter tank, dried, sampled, and packed for shipment.

A lead-pipe air lift was used to circulate the liquid in the refining box.

The solutions were finally passed through zinc-boxes after from 80 to 90 per cent. of the gold had been precipitated electrically.

At this plant it was found that for each pound of copper and gold precipitated a little over 1 lb. of cyanide was regenerated in the electrical boxes.

Figs. 48, 49, and 50 show diagrammatically some suitable arrangements for the electrical connections.

**Precipitation without Separation Solution from Ore.**—Methods with this object have been devised by Rae, Wanliss and Julian, Pelatan and Clerici, Mumford, and others. In each case the method consists of the electrical precipitation of the gold on a copper or amalgamated copper surface.

In the Mumford method, as applied to West Australian slimes, a revolving cylinder was used. It was found that with a stationary cylinder, and when the current exceeded 0.8 ampere per square foot, the gold precipitated from a

<sup>1</sup> *Eng. and Min. Jour.*, March 16, 1907.



6-dwt. solution faster than it amalgamated. Rotation of the cylinder hastened amalgamation.

Rotating iron anodes in a stationary tank were used in the Pelatan-Clerici

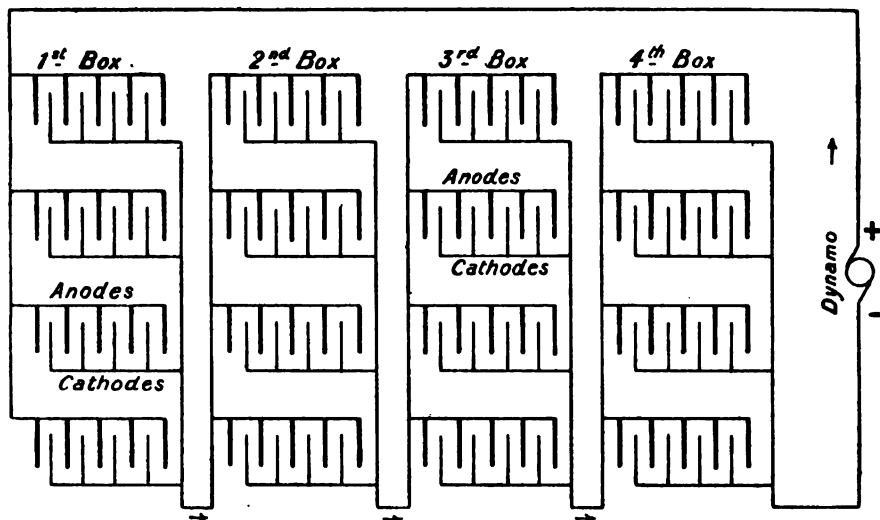


FIG. 48.—Arrangement of Circuit with Electrodes in Parallel and Boxes in Series.

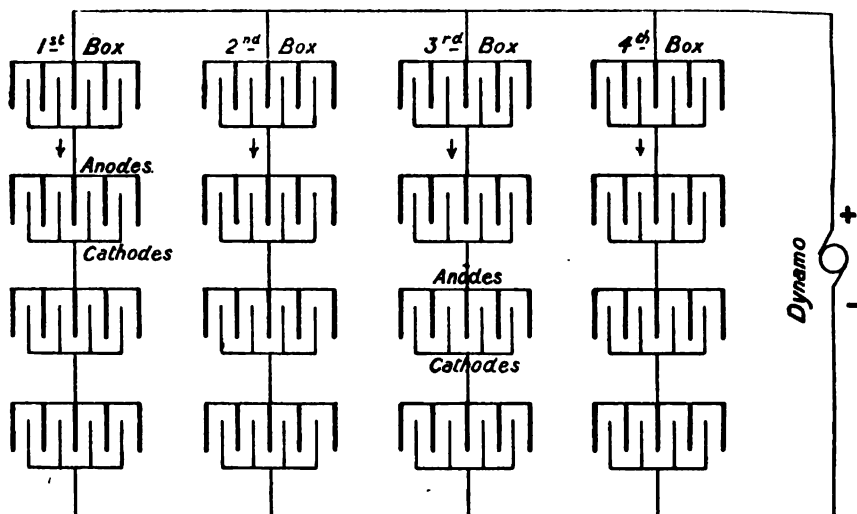


FIG. 49.—Arrangement of Circuit with Electrodes in Series and Boxes in Parallel.

process. From 4 to 20 lb. of salt per ton of ore was added to the pulp in order to precipitate sodium into the mercury on the copper cathode.

Neither of these methods have survived to the present day.

**Pfleger's Process.**—This was developed by Julian and worked on a full-sized scale for three months with remarkably good results. With original

solutions carrying from 3 dwt. to 1 oz. per ton, the residual solution consistently assayed from a trace to a few grains.

The process consists in employing a short-circuited simple cell, having zinc anodes in alkali, and iron cathodes in gold-bearing solution, the two solutions being separated by a porous partition. The working compartment was a long narrow-compartment box, each compartment being divided by porous partitions into a number of longitudinal divisions. The divisions through which the solution flowed were 1 in. wide and contained the cathodes. The

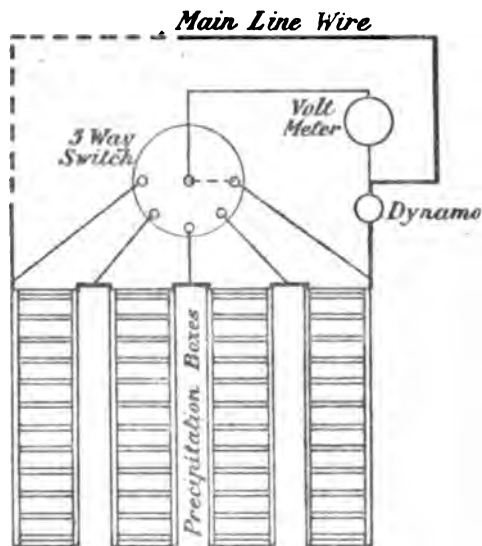


FIG. 50.—Arrangement of Switch.

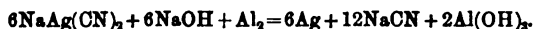
alternate divisions were 3 in. wide and contained bars of zinc in a 1 to 5 per cent. caustic-alkali solution. Each cathode was built up of five sheets of 12-mesh iron-wire gauze, placed parallel to each other and  $\frac{1}{8}$  in. apart. The zinc anode was short-circuited to the cathode, and the gold solution flowed through the cathode divisions. A cloth made from hard-spun ramie fibre, woven fairly closely, answered perfectly for the porous partitions.

The practical difficulties met with in working on a large scale were, however, insuperable.

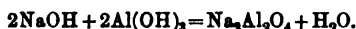
## CHAPTER XXXI.

### PRECIPITATION BY ALUMINIUM, CHARCOAL, AND SODIUM SULPHIDE.

**Precipitation by Aluminium.**—The use of this precipitant was patented by Moldenhauer in 1893. The advantage of aluminium over zinc is that the former, when used as a precipitant for metal in cyanide solutions, does not form a cyanide-precipitant compound. During precipitation of the metals the aluminium is converted into the hydrate, the reaction being expressed by Roessler thus:

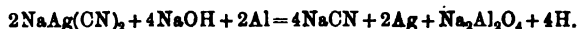


The aluminium hydrate dissolves in excess of sodium hydrate, forming sodium aluminate, thus:



According to these equations one part of aluminium should precipitate about twelve parts of silver.

E. M. Hamilton suggests<sup>1</sup> the following equation as representing the reactions occurring in the case of a low-grade solution with caustic soda and aluminium in excess—



According to this equation one part of aluminium precipitates about eight times its weight of silver, and hydrogen is liberated during the reaction.

**Precipitation with Aluminium Plates and Shavings.**—A process for the precipitation of gold and silver from cyanide solutions was patented in 1893 by Carl Moldenhauer. In the same year Julian conducted experiments on a large scale at the Rand Central Ore Reduction plant on behalf of the Deutsche Gold und Silber Schiede Anstalt, using this process for precipitating instead of zinc.

The aluminium was first used in the form of plates, placed vertically at a distance of  $\frac{3}{4}$  in. apart, in a precipitating box divided into compartments, similar to that used in electrical precipitation. The result was, however, unsatisfactory, as a 6-dwt. solution at the head of the box gave a 2-dwt. sump solution. Aluminium shavings were next tried, when the resulting sump solution was lowered to about 10 or 12 grains.

It is necessary for efficient precipitation that either free alkali or free acid should be present in the solution. For obvious reasons the latter is difficult to

<sup>1</sup> *Eng. and Min. Jour.*, May 10, 1913.

apply, if not impracticable, in the cyanide process. It was found after the process had been at work some weeks that alumina began to deposit on the shaving, which retarded precipitation of the gold. This was easily rectified by the addition of a little alkali at the head of the box, but as time went on the ever-increasing quantity of alumina in the working solutions became more and more troublesome. Attempts were made to remove the alumina by precipitating it in the solution tanks, and then leave it behind in the ore, but it was found that whenever alumina was precipitated in a cyanide solution it also carried down a large proportion of the free cyanide.

**Precipitation with Aluminium Dust.**—The increasing recognition of the fact that the use of zinc causes a fouling of the cyanide solution has led, in several cases, to the adoption of aluminium as a substitute. The aluminium is usually employed in the form of dust, so that the quantity used and the time of contact with the solution can both be controlled and limited to the minimum necessary for efficient precipitation.

**Application at Deloro.**—At Deloro, Ontario, it was found that the consumption of cyanide, due to the solution of zinc in this solvent, was abnormally high. Foul solutions and a low-grade bullion resulted. S. F. Kirkpatrick therefore devised a method for the use of aluminium dust, which was introduced on a commercial scale in 1908, and was responsible for the precipitation of over 14,000,000 oz. of silver up to the middle of 1913. The following description of method and equipment is condensed from an article by Kirkpatrick:<sup>1</sup>

The emulsifying and precipitation vat is fitted with a vertical shaft carrying a number of horizontal propeller-blades arranged to circulate the solution and to produce a vortex which sucks down the light aluminium powder, thus ensuring a thorough contact between solution and precipitant. A three-throw pump, filter-presses, and the necessary solution tanks complete the essential apparatus.

From 20 to 30 tons of pregnant solution is run into the precipitation tank. A sample drawn from this is first titrated for free cyanide. Aluminium powder and caustic soda, in slight excess, are then added to from 200 to 300 c.c. of the same solution. This is then well shaken in a bottle for two minutes, filtered, and the clear solution again tested for free cyanide. From the difference between the two titrations an approximation of the amount of silver present in the tank is calculated.

An amount of aluminium dust, slightly greater than one-eighth of the approximate weight of silver present, together with caustic soda as required, is then added to the precipitation tank. The quantity of precipitant used is in excess of the amount indicated as required by Roessler's equation, as might be expected with complex solutions. Kirkpatrick agrees, however, that the precipitation may take place partly according to Hamilton's equation.

After sufficient agitation the precipitate pulp is transferred to one of

<sup>1</sup> *Eng. and Min. Jour.*, June 28, 1913.

the presses by means of the 3-throw pump. The precipitate in the press is then water washed, air dried, and discharged.

The flux used in smelting is as follows :

|                           |     |
|---------------------------|-----|
| Precipitate, . . . . .    | 100 |
| Borax, . . . . .          | 16  |
| Sodium nitrate, . . . . . | 8   |
| Fluorspar, . . . . .      | 4   |
| Sand, . . . . .           | 5   |

The bullion produced averages about 999 fine.

**Application at Nipissing Mining Co., Cobalt.**—At this plant E. M. Hamilton<sup>1</sup> found that the use of zinc fouled the solution to an extent which made its continued use a practical impossibility. The consumption of cyanide was also high and precipitation unsatisfactory, due, apparently, to the quantity of arsenic in the ore. These troubles were overcome by the use of aluminium dust; and, in addition, economies of operation were effected.

At this plant the pregnant solution passes from the clarifying filter to a storage vat, and thence by launder to one of two small mechanical-agitation tanks, 5 ft. diameter by 6 ft. deep, placed in tandem. The solution passes from the bottom of the first agitator to the top of the second; and from the bottom of the second it is pumped to the precipitate press. The stirrers consist of vertically placed planks attached to two horizontal spiders, one near the bottom and one near the top of a central vertical shaft which is rotated at about 60 revolutions per minute.

The level of the solution in the second agitator is automatically kept constant by a float, which controls a valve in a bypass provided in the pump circuit.

The supply of dust is regulated by sampling the effluent from the press and testing this with a little strong sodium sulphide solution. If the faintest colouration occurs, then the feed of aluminium dust is increased until the test shows complete precipitation.

At this plant the aluminium consumed is one-third of the weight of silver recovered.

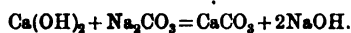
**Limitations in the Use of Aluminium Dust as a Precipitant.**—Up to date no serious effort has been made to use aluminium in place of zinc as a precipitant in the cyanide treatment of gold ores, although it is equally efficacious as a precipitant with gold as with silver solutions. Its introduction in this connection will doubtless be only a matter of time as soon as the difficulties now apparent have been successfully surmounted.

In connection with the use of aluminium as a precipitant E. M. Hamilton<sup>2</sup> has pointed out that the addition of lime in the treatment of the ore should be restricted to a minimum because any excess will form an insoluble precipitate of calcium aluminate, which will tend to reduce the grade of the resulting precipitate below an economic limit. Caustic soda is used as a neutraliser and must be present in excess to ensure efficient precipitation.

<sup>1</sup> *Eng. and Min. Jour.*, May 10, 1918.

<sup>2</sup> *Loc. cit.*

At the Butters Divisadero mine plant, the solution is treated, before precipitation, with soda ash. Any lime present is thrown down as carbonate, and caustic soda is produced, according to the following equation :—



#### **Advantages of Aluminium Precipitation :**

- (i) Absence of fouling of the solution with cyanide compounds of the precipitant.
- (ii) Complete regeneration of cyanide during precipitation.
- (iii) Reduced precipitation costs.

**Molloy's Process.**—In this method the gold-bearing solution was passed over a surface of mercury in which sodium was continuously deposited. The apparatus consisted of a large shallow tray covered with mercury. A central narrow longitudinal compartment was formed in this by two partitions which dipped into the mercury. In this compartment a strong solution of sodium carbonate was placed. This was electrolysed, the mercury at the bottom being the cathode, with anodes of peroxidised lead. As the sodium deposited in this central compartment it diffused to the mercury in the outer compartment, through which the gold-bearing solution constantly flowed; and the gold was precipitated into the mercury by the action of the sodium amalgam.

**Precipitation by Charcoal.**—According to the laws of electro-chemical action, already discussed, carbon *per se* cannot precipitate gold or silver, as it is strongly electro-negative to each of them in a cyanide solution. That charcoal is able to abstract gold or, rather, a gold compound from auro-cyanide solutions was, however, realised in the early days of the industry.

In the colony of Victoria, where this method has been most frequently used, the practical procedure was as follows: The charcoal is crushed through a small pair of toothed rolls; the fine stuff is separated by sieving and thrown away, the harder portions are eliminated by washing, and the remainder is put into small tubs about 2 ft. in diameter inside and slightly deeper. These are placed on the floor of the precipitation house in from three to six parallel rows, each row being a few inches lower than the preceding one. Each tub has an earthenware pipe about 5 in. in diameter placed vertically in the centre of it, also filled with charcoal. The solution to be precipitated is divided equally between the tubs in the top row, and from each of these it flows successively through the tubs below, downwards through the central pipe of each, and upwards through the outer charcoal in the tub. A lid is provided over this, to prevent the pieces of charcoal from floating on the liquor and passing away.

At suitable intervals the top tubs are removed for cleaning-up, and each row of tubs is then transferred to the next higher position, while a set containing fresh charcoal is placed in the lowest row. Hence the custom of using such small tubs. It is evident that a considerable amount of labour

is required for this purpose, for in a plant treating 84 tons of solution daily there are, according to John I. Lowles,<sup>1</sup> 198 tubs, of which at least 48 are moved every day, this requiring two men.

The charcoal is preferably burnt in a Turnbull furnace. This has a conical combustion chamber, having a cover at the top provided with a feed door and a small grate near the bottom, and below the fire-bars is an ash-pit which can be locked up. The flue from the top of the combustion chamber terminates under water in a separate compartment which has an outlet above the water level, provided with a steam ejector or other apparatus to maintain a draught. Thus all the products of combustion are caused to pass into and through the water, with a view to preventing any loss of gold by volatilisation or dusting. A small fire is lighted in the grate at the beginning of the operation, but afterwards the charcoal continues to burn without assistance, fresh supplies being fed in by the door in the cover as required.

In order to facilitate comparisons of this method of precipitation with others, the following figures have been calculated from published data of various plants: Cubic feet of charcoal required in tubs per ton of solution treated daily, 7 to 14. Specific rate of flow, from 0.016 to 0.024 cubic foot per minute for each square foot of sectional area.

Conversely, each ton of solution per day requires from 0.9 to 1.4 square feet of section in the column of charcoal.

Length of column, 6 to 12 ft., *i.e.* from three to six tubs in series.

**Recent Advance in Charcoal Precipitation.**—The progress in the utilisation of charcoal as a practical precipitant of gold from aurocyanide solutions has been summarised,<sup>2</sup> and attention drawn to the fact that the use of charcoal in this connection has, hitherto, never reached the point of importance where it might be considered as a serious rival to zinc. The only exception to this statement may be found in the case where gold from waste solution discharged from a cyanide plant to reduce fouling, or to prevent undue accumulation of liquor, is recovered by this method. Appreciable quantities of gold may also be saved, by the use of charcoal, from decanted solution from slime-tailing dams, or where the seepage from tailing-filled stopes carries small amounts of dissolved metal. In such instances the results may be termed satisfactory only inasmuch as an amount of gold is recovered that would otherwise be lost. Effort is seldom made to effect a complete saving, the equipment is usually neglected during the month and attention only paid to it when the urgent need of an addition to the gold yield is in evidence.

On a commercial scale the substitution of lump charcoal for zinc could not be considered satisfactory from any standpoint. A large amount of charcoal is needed to recover a small amount of gold. The product, even after burning, is bulky; and the gold content of the ash to be smelted is exceedingly low. The efficiency of the process, unless an enormous amount of charcoal is

<sup>1</sup> *Trans. Inst. Min. and Met.*, vol. vii. p. 192.

<sup>2</sup> "Charcoal Precipitation of Aurocyanide," by A. W. Allen, *Met. and Chem. Eng.*, June 15, 1918.

used, is unsatisfactory. No progress in the adaptation of charcoal as a means of recovering gold from cyanide solution has been made for many years; and its use in this connection has been considered as unfeasible as the cyanide process itself was viewed before MacArthur and Forrest invented a method of procedure capable of practical application.

**The Work of Moore and Edmands.**—A noteworthy advance is now due to the efforts and ingenuity of K. Byron Moore and H. R. Edmands who have resurrected the question and, after exhaustive laboratory experiments, have perfected a system of precipitation with charcoal slime which apparently leaves little to be desired. The method has been finally adopted in Western Australia and South Africa. In the former case a foul and complicated cyanide solution containing antimony and arsenic is successfully freed of its gold. With zinc precipitation there was trouble and loss from high tail solution, the cost of zinc was considerable, and even after extensive refining, the bullion was comparatively low grade. By the adoption of the new charcoal method the effluent from the extractors may now be reduced to contain but a trace of gold. The bullion, after simple smelting, is valued at about \$20 per oz., as compared with about \$17 per oz. for a similar product after zinc precipitation. The saving in the cost of recovery of the gold is about 25 cents per ton of ore treated.

After a year of operation a statement of result was published by H. G. Walton.<sup>1</sup> No fouling agent similar to the double zinc cyanide appears in the solution; and such a result would automatically effect a saving in cyanide consequent on a permissible reduction of cyanide strength in the solution. No figures are available in this connection, but in a later publication by H. R. Edmands<sup>2</sup> it is to be noted from comparative results of the use of zinc dust and charcoal that the necessary cyanide strength in the case of zinc was five times as much as when charcoal was used.

**The Moore-Edmands Process.**—The principal features of the new Moore-Edmands process are as follows: The charcoal used is water-quenched when red-hot, washed, and wet ground in a small ball-mill to as fine a product as possible. A special form of extractor is used, consisting of a miniature vacuum-filter of (what is known outside Australia as) the Butters type. The suspended frames are 2 ft. 3 in. by 2 ft. 9 in., and there are twenty-one to each extractor. The filter media are, respectively, coconut matting as a support, filter canvas, and an outer protective layer of calico. The extractor contains four agitators in the bottom of the hopper, attached to a horizontal shaft.

**Method of Operation.**—Solution and charcoal pulp are pumped to an extractor and the vacuum started as soon as the leaves are properly submerged. Fresh solution is added concurrently with the formation and maintenance of cake, and the supply of liquor is such that a continuous stream passes through the charcoal. The propellers are kept in operation and percolation continued for 24 hours, or some other pre-determined time. The vacuum is then released,

<sup>1</sup> *Report of the Department of Mines of West Australia*, 1916.

<sup>2</sup> *Bull. Inst. Min. and Met.*, London, March 1918.



the charcoal cakes dropped to the hopper bottom of the extractor, emulsified, and reformed on the filter leaves by the application of vacuum. This cycle of operations is continued at intervals so long as the charcoal possesses sufficient activity to precipitate the gold. The extractors are worked in series, the one whose cakes are approaching gold saturation receiving the richest solution, the tailing from this going to the second extractor, and being finally impoverished by contact with fresh charcoal.

The final emulsified charcoal pulp, loaded with gold, is delivered to a clean-up press, washed, and dried with compressed air.

Burning of the charcoal is carried out in braziers, 18 in. by 18 in. by 18 in., and made of  $\frac{1}{2}$ -in. aperture stout wire screening. The operation is carried out in a locked steel chamber and takes about 48 hours.

The flux adopted is as follows :

|                        |            |
|------------------------|------------|
| Precipitate, . . . . . | 100 parts. |
| Borax, . . . . .       | 40 to 50 „ |
| Sand, . . . . .        | 50 „       |
| Salt, . . . . .        | 10 „       |

Plain graphite crucibles are used and have a long life.

A comparison of results with charcoal and zinc precipitation is as follows :

|                   | Charcoal.   | Zinc.       |
|-------------------|-------------|-------------|
| Gold, . . . . .   | 954.4 fine. | 825.2 fine. |
| Silver, . . . . . | 7.0         | 21.9        |
| Base, . . . . .   | 38.6        | 152.9       |

**Amount of Charcoal needed.**—Walton states that 1 ton of charcoal will precipitate 770 oz. gold by this method, and that the weight of calcined ash is about one-third of the original weight of the charcoal. Edmands gives the percentage of gold in the calcined ash as from 10 to 14 per cent. The enormous increase in the precipitating power of charcoal by fine grinding is indicated by the fact that, by the old method of percolation through extractor boxes, 5 tons of charcoal was needed to precipitate about 700 oz. gold.

The charcoal used at the Yuanmi mine is an otherwise waste product from producer-gas manufacture. The wood used is locally known as mulga scrub and is a variety of the acacia family which includes several wattles indigenous to Australia. It is possible that the character of the wood, which in some species contains a considerable amount of tannin, has a favourable effect on the final result. Different woods have widely different precipitating values when made into charcoal and used for the recovery of gold from cyanide solutions.

The sole drawback to the method appears to be the amount of material to be smelted. The actual smelting entails no difficulties and there is little corrosion of crucible. The latter might even be reduced by an alternative flux. In medium- or large-sized plants such low-grade precipitate is best smelted in large tilting furnaces—an arrangement permitting the periodic pouring of slag only, the gold being allowed to accumulate in the bottom of the crucible, where it refines. This method minimises the temporary loss by

shotting met with when small charges of gold are poured through a large bulk of cooling slag. The furnace also acts as a safe during night operations, when there is less available supervision, and when minimum handling of bullion is advisable.

**Theory of Charcoal Precipitation.**—The various theories propounded to account for the action are as follows:

*Occluded Oxygen Theory.*—It has been suggested that an insoluble gold cyanide and sodium cyanate is formed.

*Electro-Deposition Theory.*—This theory assumes that gold is deposited as metal as the result of electrical action. It is disproved by the fact that there is no corresponding liberation of cyanide during precipitation.

*Carbon Monoxide Theory.*—The assumption that a chemical action takes place between the occluded carbon monoxide gas and the aurocyanide whereby a carbonyl aurocyanide, or other analogous compound, is formed has been widely accepted. Criticism of this theory must be mainly constructive and in opposition. All the assumptions are hypothetical and no parallel action is known with charcoal.

*Adsorption.*—The theory that the precipitation is due to a simple adsorption of aurocyanide— $\text{KAu}(\text{CN})_2$ —is supported by the following facts:

1. The loss of cyanide, by titration, agrees with theoretical considerations.
2. Precipitation efficiency varies inversely with the concentration of the solution, and a true adsorption curve may be plotted from actual results.
3. Precipitation is proportionate to surface exposed, finer grinding giving better results.
4. Methods of preparation of the charcoal which drive off occluded gas (quenching when red-hot and wet grinding), and so increase surface energy of the charcoal, result in a more favourable action.
5. The addition of acid accelerates precipitation, whereas alkali retards it. This is a parallel instance of the obstructive effects of hydroxyl as compared with hydrogen ions as demonstrated by Lachs and Michaelis in connection with the adsorption of an aqueous solution of potassium chloride by charcoal.
6. The adsorbed gold compound is insoluble in ordinary solvents although released by the addition of a solution of different surface tension, such as a soluble sulphide—a parallel to the case where picric acid adsorbed on platinum may be released by alcohol, although unaffected by water, its natural solvent.

**Precipitation with Sodium Sulphide.**—An interesting and important contribution to the literature of the subject has been made by R. B. Watson, who describes<sup>1</sup> the practical application of sodium sulphide, as a precipitant of silver from cyanide solution, at the Nipissing plant. The following details of theory and practice are abstracted from the article in question. E. M. Hamilton,<sup>2</sup> while admitting that sodium sulphide would prove a suitable precipitant of silver in cyanide solutions, suggested that two objections are apparent—(1) complete precipitation is not obtained without using an excess of the precipitant, which would then have to be removed by a lead compound,

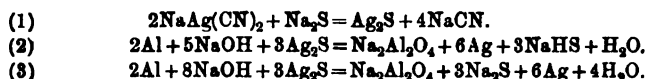
<sup>1</sup> *Trans. Can. Inst. Min. Eng.*, 1917.

<sup>2</sup> *Eng. and Min. Jour.*, May 10, 1913.

the alternative being a tailing solution too high in silver to form an effective residue wash; and (2) the silver would be obtained in sulphide instead of metallic form. That these two views were erroneous has been demonstrated by subsequent practice.

It was found that the solution could be precipitated to 0.2 oz. silver without an excess of sodium sulphide in the barren solution. The second objection was met by converting the silver sulphide to bullion by using the same principle employed in the desulphurisation of the raw ore before it is cyanided. This principle is that the various sulphide combinations of silver are decomposed by contact with metallic aluminium in a caustic-soda solution. This is an important feature of the process, resulting in the production of fine bullion at low cost.

In the sodium hyposulphite process, before silver ores were treated by cyanide, the treatment and marketing of the sulphides produced was the principal obstacle. Various methods were devised for the treatment of this product at the plant, but all were costly and unsatisfactory. The best of them, however, was the Dewey-Walter process, in use at the Marsac mill, Park City, Utah. The reactions involved in this process are as follows:



The commercial sodium sulphide used is supposed to be  $\text{Na}_2\text{S} + 9\text{H}_2\text{O}$ , which would be equivalent to 32.5 per cent.  $\text{Na}_2\text{S}$ . A concentrated form of sodium sulphide can be obtained.

By the use of equation (1) it is found that it would require 0.076 lb. avoirdupois of commercial sodium sulphide to precipitate 1 oz. troy of silver. In practice the amount required is 0.12 lb. In the desulphurising operation as shown in equation (2) one troy ounce of silver requires 0.021 lb. of NaOH. This would be 0.028 lb. of commercial caustic soda carrying 76 per cent. NaOH. If an excess of caustic is present in the operation, the reaction in equation (3) may take place, in which case 0.034 lb. of NaOH is used for each ounce of silver desulphurised. To prevent this taking place, the amount of silver present in the charge to be desulphurised is estimated as closely as possible and 0.03 lb. of commercial caustic soda is added for each ounce of silver contained. The theoretical amount of aluminium necessary to desulphurise 1 oz. of silver is 0.0057 lb. avoirdupois, the amount actually used in practice being 0.006 lb.

**Practical Use of Sodium Sulphide.**—The process is carried out by dissolving the sodium sulphide in an iron barrel 2 ft. 6 in. diameter by 4 ft. long, into which is flowing constantly a small stream of barren solution from the filter press. The barrel is divided by a vertical partition reaching nearly to the bottom. There is an excess of sodium sulphide in the barrel, and the overflow pipe of the opposite side of the partition from the inflow carries the precipitant to a tank 5 ft. diameter by 6 ft. deep, where it is agitated with pregnant solution, which is then pumped to the filter press.

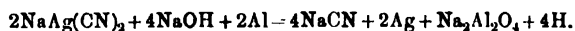
The amount of sodium sulphide added is regulated by a boy, who constantly tests the inflowing barren solution with pregnant solution and with sodium sulphides, to determine whether there is an excess of sodium sulphide or unprecipitated silver present in the discharge from the press. The amount of barren solution coming into the barrel, and consequently the overflow of sodium sulphide, is regulated by a valve on the intake pipe in front of the operator.

The solution from the filter press is aerated thoroughly in the barren-solution tank by air lift and circulating pump, the object being to oxidise any sodium sulphide should there happen to be a slight excess in the precipitated solution. The shiftman always tests the solution for soluble sulphide before running it to the treatment tank, and no bad effect has been noticed from this cause. Aeration is used in preference to lead salt, as lead has a bad effect on extraction. A pressful of sulphide precipitates contains 25,000 oz. silver. This is drawn into a tank 7 ft. diameter by 5 ft. deep, and the proper amount of caustic soda is added, 0.03 lb. for each ounce of silver. The dilution is four parts of solution to one of sulphide, and the strength of the solution in NaOH is about 8 per cent. The pulp in this tank is kept mixed by the usual mechanical stirring device. Over the tank is mounted a small tube mill 15 in. diameter by 6 ft. long, which makes 10 r.p.m. The charge of aluminium ingots weighs 300 lb. and fills the tube mill half full. Below the tank is a centrifugal pump which circulates the pulp from the tank through the tube mill for 10 or 12 hours.

When desulphurisation is almost complete, the precipitate rubbed on a piece of paper by a spatula gives a silvery colour and a metallic lustre. It is then pumped to another press, where it is washed for 2 hours; the resulting product, then mixed with a small amount of water, is charged, without drying, to the reverberatory refining furnace and melted down to bullion, 996 fine. The slag is put through a small blast furnace.

The grade of the precipitate depends largely on how well the pregnant solution has been clarified before precipitation, but under ordinary conditions it assays about 23,500 oz. before and 26,500 oz. after desulphurisation. The final product carries from  $\frac{1}{2}$  per cent. to 1 per cent. sulphur. Gold is not precipitated in an alkaline cyanide solution, and copper does not come down if there is sufficient free cyanide in the solution.

It has been shown that a large part of the saving in the use of aluminium over zinc dust is in the regeneration of cyanide. By comparing the first equation with Hamilton's equation, given here, it will be seen that the regeneration is the same when sodium sulphide is used.



The process as described is limited in its application to silver ores, as gold sulphide is not precipitated. It is possible, however, that the procedure given could be used to precipitate the silver, followed by the precipitation of the gold in the solution by aluminium.

**Use of Sodium Sulphide in West Africa.**—A condensed account of the use of sodium sulphide as a corrective after the premature precipitation of gold by graphite appears in the annual review number of *The Engineering and Mining Journal*, Jan. 11, 1919,<sup>1</sup> in which it is shown that an interesting development in the metallurgy of gold has recently taken place in connection with the cyanidation of the quartz ore at Prestea mine, Gold Coast Colony, West Africa.<sup>2</sup> The deposit is crossed with seams of carbonaceous schist, which is also found in close proximity to the vein; and, during cyaniding operations, some of the gold is prematurely precipitated on the graphite. This resulted in high residues; but subsequent experimentation indicated the feasibility of using sodium sulphide as a corrective in a later stage of treatment.

As practised at the Prestea, the new modification consists of drainage, after cyanide treatment in leaching vats, and retreatment with 0.2 per cent. sodium sulphide solution. The material is then water washed, and the residue discharged. The gold-sodium sulphide solution is passed through copper shavings, where the precious metal is deposited, and receptacles similar to ordinary zinc boxes are used. The gold precipitate is filter-pressed, dried, and smelted. Copper-gold bullion, matte, and slag result, which assay 100 fine, 33 oz. per ton, and 12 oz. per ton respectively. The new process recovers an additional 50 cents' worth of gold per ton of sand treated.

With regard to the theory of this interesting development, it has been proposed<sup>3</sup> that the sulphide was a solvent for the gold precipitated on the carbon. More recently a hypothesis has been advanced<sup>4</sup> which draws attention to the evidence in favour of the theory that a gold solution is adsorbed by the carbon rather than chemically acted upon by occluded gas, and that the addition of sulphide introduces new factors of surface-tension influence, with the result that a part of the adsorbed gold compound is released and is available for treatment with a precipitant.

<sup>1</sup> "Metallurgy of Gold and Silver," by A. W. Allen.

<sup>2</sup> "The Sodium Sulphide Process of Gold Recovery," by F. Wartenweiler, *Trans. I.M.M.*, 1918.

<sup>3</sup> "The Precipitating Action of Carbon in Contact with Auriferous Cyanide Solutions," by W. R. Feldtmann, *Trans. I.M.M.*, vol. xxiv. p. 329.

<sup>4</sup> "Charcoal Precipitation of Aurocyanide," by A. W. Allen, *Met. and Chem. Eng.*, June 15, 1918, p. 644.

## CHAPTER XXXII.

### REFINING OF PRECIPITATION PRODUCTS.

THE mode of procedure subsequently adopted, to refine the products resulting from the use of zinc as a precipitant, depends on the class of material being treated, the richness of the solution from which the metal has been precipitated, and whether gold or silver is in predominant quantity.

**Silver Precipitate from Cyanide Solutions.**—The precipitate in this case is generally free and easily screened. The mesh used usually varies from 900 to 1200 holes per square inch, and the washed oversize is returned to the boxes. The precipitate usually carries a high percentage of silver, and no further treatment beyond filter-pressing is necessary before the material is smelted.

**Gold Precipitate from Rich Solution.**—In some instances it has been found practicable and economical to remove only the fine precipitate from the boxes, smelting this direct after screening and filter-pressing, as with silver precipitate. Such precipitate is, however, of much lower percentage grade than the corresponding silver precipitate; and a further disadvantage arises from the fact that metal of considerable total value must necessarily be left in the boxes and carried over after the clean-up.

**Gold Precipitate from Treatment of Normal Grade Ore.**—The short zinc, together with the filter-pressed gold slime, will require further treatment before being in a condition suitable for ordinary smelting. A choice of the following methods can be made:

- (1) Roasting with or without nitre.
- (2) Sulphuric acid treatment.
- (3) Bisulphate of soda treatment.
- (4) Smelting direct with litharge and cupelling the resultant base bullion.

(1) **Roasting.**—In this method the zinc shorts and gold slime are either roasted direct or are previously sprinkled with a brine of nitre, the amount used varying from 3 to 10 per cent. of the weight of the wet product from the boxes. The mixture is placed on trays in a muffle furnace and the heat raised so that the zinc is oxidised. The oxide fumes given off are drawn away to a chimney by means of a carefully regulated draught.

A modification of this method was recommended by W. A. Caldecott<sup>1</sup>

<sup>1</sup> *Jour. Soc. Chem. Ind.*, xvii. p. 3, 1897.

in 1897. The slime is dried nearly to dusting, then well mixed with silica and nitre in the following proportions :

|                           |           |
|---------------------------|-----------|
| Slime, . . . . .          | 100 parts |
| Sand, . . . . .           | 30 "      |
| Powdered nitre, . . . . . | 30 "      |

The mixture is spread on iron trays to a depth of about  $\frac{3}{4}$  in. and heated to a little over the boiling point of water. Then a live coal and a pinch of nitre is applied to the mixture, when combustion spreads throughout the whole layer and proceeds quietly, but intensely, giving off dense fumes of zinc oxide, and leaving behind a coherent non-dusting product.

The action is said to consist in the oxidation of a portion of the carbonaceous matter and base metals, and a partial combination with the sand, to form a soluble silicate. The following is the flux used in smelting the product :

|                                 |           |
|---------------------------------|-----------|
| Oxidised slime, . . . . .       | 100 parts |
| Anhydrous borax, . . . . .      | 40 "      |
| Dry sodium carbonate, . . . . . | 10 "      |

This is said to give a good gold-free slag, and bullion about 800 fine.

In either of the above methods of eliminating zinc by roasting there is a possibility of gold being carried away with the zinc fumes. Mechanical loss may also occur at various stages, due to strong cooling currents. It is practically impossible to determine these losses with any accuracy on account of the extreme difficulty of obtaining average samples of the material before and after treatment.

(2) **Sulphuric Acid Treatment.**—The use of sulphuric acid constitutes the general method for the elimination of zinc from extractor-box gold slime. The zinc sludge is transferred, together with the screened filter-press product, to a specially designed clean-up vat, which is preferably lined with lead sheeting. The weak acid in the vat should be made up with hot water so that the mixture carries from 10 to 15 per cent. of concentrated sulphuric acid. The zinc slimes are added at intervals, care being taken to avoid the ill-effects of violent preliminary action. Naked lights should be kept away from the neighbourhood of the vat ; and inhalation of the fumes, which are invariably poisonous, prevented as much as possible by the provision of a hood which is lowered over the vat immediately after the addition of each tray of zinc or bucket of acid. If possible, the vat should be provided with an agitation apparatus, in which case the zinc is added almost continuously through a hopper.

Agitation is continued after the vat has received its complete charge of slime. Hot water and more acid are then added from time to time and as long as there is continued evolution of hydrogen. The secret of success in sulphuric acid treatment lies in the provision of an ample hot-water supply. This hot water should be added to the charge before each addition of acid.

When there is no further action to be observed on the addition of fresh

acid the vat should be filled with hot water, the precipitate allowed to settle, the zinc sulphate solution pumped through the clean-up press, and the operation repeated. By this means the bulk of the zinc will be eliminated before the precipitate enters the press, which is then filled in the usual manner. Agitation of the gold-slime pulp is continued in the vat during pumping, thus ensuring a homogeneous cake in the frames of the press. As soon as the press is filled, the cakes are washed with hot water, either from the high-pressure plates or by the median system, air or steam dried, and discharged by hand.

(3) **Bisulphate of Soda Treatment.**—This is a modification of the sulphuric acid treatment, which has been largely adopted on the Rand during recent years. The use of bisulphate is said to result in an economy of operation, but the application of the solvent solution and the operation of cleaning up generally is the same as is adopted with sulphuric acid. A much more extensive refining plant is required in the case of the bisulphate, including means for handling, breaking, and dissolving the salt. The chief reason for its use on the Rand is its comparatively low price as a byproduct of the local dynamite factory.

(4) **Tavener Process of Smelting with Litharge.**—Another method of dealing with the precipitate consists essentially in smelting it with litharge in a pan furnace and afterwards cupelling the lead bullion thus produced. This neat adaptation of the scorification assay method, for use with large quantities of material, was introduced and developed at the Bonanza mine, Johannesburg, by P. S. Tavener.

The sludge is pumped from the clean-up vat to the filter press and made into cakes as usual, and the fine zinc remaining in the vat is collected separately. The two products are separately dried on trays in an oven for not more than fifteen minutes, care being taken in each case that the mass is still coherent when removed from the oven. The slime cakes are then rubbed through a 4-mesh sieve, roughly weighed, and mixed with flux. The mixture is passed through a sieve and shovelled into the smelting furnace. Then the fine zinc is treated in the same manner and charged into the furnace above the slime, with the intention of preventing the latter from dusting.

The furnace charge is banked up towards the centre and lightly covered, first, with litharge, and, secondly, with readily fusible slag.

The fire is then lighted, and about two hours later the heat may be considerably increased and a high temperature maintained. When the charge is molten, any byproducts, such as sweepings, old slag, or almost anything which contains gold, may be added to the bath. The charge is then well stirred, and sawdust added to reduce any remaining litharge until the slag gives a clean sample on a rabble. The slag is next run into pots in the usual manner as far as possible, and most of the remainder removed with the rabble. The fire door is opened and lime is thrown in to thicken the last of the slag, which is afterwards removed and kept for resmelting.



After sampling with a ladle, the clean lead bullion is tapped and run into ingot moulds in the ordinary way.

The next step is to cupel the lead bullion on a bone-ash test, the usual practice being followed. The gold is allowed to partly cool; it is then broken in half and drawn out of the furnace. It is next broken up into smaller pieces of suitable size for resmelting in crucibles.

The exact fluxing mixture varies with the composition of the sludge, and may be ascertained by small trials in the assay office; but, generally speaking, less slag is needed in the large furnace than in crucible tests. The quantity of litharge depends chiefly upon the amount of gold present, as it is said to be desirable to limit the value of the lead bullion to 8 or 10 per cent. of gold.

The following figures give the proportions hitherto found suitable:

|   |                                     |
|---|-------------------------------------|
| (1) Slime, partly dried, . . . . .        | 100 parts by weight.                |
| PbO, . . . . .                            | 40 to 60 " "                        |
| Assay slag, . . . . .                     | 10 to 15 " "                        |
| Slag previously used, . . . . .           | 10 to 15 " "                        |
| Sawdust, . . . . .                        | $\frac{1}{2}$ to $1\frac{1}{2}$ " " |
| (2) Fine zinc, partly dried, . . . . .    | 100 parts by weight.                |
| PbO, . . . . .                            | 100 to 150 " "                      |
| Mixed slag as above, . . . . .            | 25 to 30 " "                        |
| No reducing agent except the zinc itself. |                                     |

The figures below show actual results from four months' work at the Bonanza mine.

|   |          |
|---|----------|
|   | Troy oz. |
| Moist weight of filter-pressed slime smelted, . . . . . | 131,342  |
| Moist weight of fine zinc smelted, . . . . .            | 111,104  |
| Total weight of material treated, . . . . .             | 242,446  |
| Lead bullion obtained and cupelled, . . . . .           | 221,407  |
| Total fine gold recovered, . . . . .                    | 12,810   |

**The Refining of Zinc-Dust Precipitate.**—The finely divided condition of the excess zinc associated with the precipitated gold and silver when zinc dust is used, is an unfavourable factor affecting refining treatment. An additional disadvantage lies in the fact that zinc in such a form is more liable to become oxidised either before or during treatment. Its solution by methods wholly applicable to zinc-shaving precipitate is therefore a difficult matter. In this connection it may be noted that the economies effected by the use of dust as a precipitant in place of shavings are in several cases more than counterbalanced by the additional expense involved on account of the necessarily complicated subsequent treatment of the product resulting.

**Treatment of Zinc-Dust Silver Cyanide Precipitate.**—This material generally contains a high percentage of silver and may be smelted direct. In order to avoid the complication due to the presence of excess zinc in the precipitate, which on account of its finely divided state cannot be screened out, the use of a partial precipitation circuit, already described, will ensure

a proportion of higher-grade bullion and less trouble in realisation, as a direct result of a restricted use in the amount of precipitant.

**Treatment of Zinc-Dust Gold Cyanide Precipitate.**—Ordinary methods are seldom available for the removal of excess zinc from this product. Acid treatment is generally only partly successful, and a further refining process is necessary. The usual method is to precede what has already been described as Tavener's process with acid treatment. By a combination of these two methods of refining, the bullion is high grade. The results are, therefore, similar to those obtained on zinc-shaving precipitate after direct litharge-smelting-cupellation treatment.

## CHAPTER XXXIII.

### SMELTING OF CYANIDE PRECIPITATE AND RECOVERY OF BULLION.

**Smelting Cyanide Precipitate without Fluxes.**—This method is in practice in cases where the precipitate has resulted from the cyanidation of a high-grade silver ore or concentrate. The bullion may be practically fine silver; and the dross or skimmings are treated separately by an alternative smelting process, involving the use of coke, slag, and limestone as fluxes.

**Fluxing the Precipitate.**—The object aimed at in fluxing a raw or refined precipitate is to obtain a neutral slag, sufficiently fluid at a moderate temperature to allow the smaller gold and silver particles to agglomerate and descend by gravity. To attain this end a rough analysis of the slime would be a great help to the smelter, because the proportion of fluxes adopted should be dependent on the composition of the material to be smelted. Much has to be learnt in an empirical way, however, and the smelter must often use his own judgment on the question of fluidity, neutrality, or other controlling factor governing the efficiency of operation. In this connection it may be mentioned that two smelters may obtain equally good results although the fluxes and their proportions may vary widely.

Chas. E. Meyer<sup>1</sup> has suggested and proved by practice a method by which the proportions of borax, silica, and other ingredients to be added may be calculated when the composition of the slimes is known. In one case dealt with the slimes contained: gold, 23; SiO<sub>2</sub>, 27.6; ZnO, 36.4, and other oxides (CuO, Fe<sub>2</sub>O<sub>3</sub>, and PbO, traces), by difference 13 per cent. "The desired object aimed at in the flux was the formation of zinc subsilicate with the ZnO present, that is, having two parts of oxygen in the acid radicle to one in the base—ZnO, SiO<sub>2</sub> or ZnSiO<sub>3</sub>. By analysis, the silica found (27.6) was in about the right ratio for the formation of this compound found by calculation as follows:

$$\text{ZnO} \frac{\text{Percentage}}{\text{Molecular weight}} = \frac{36.4}{81} = .44.$$

$$\text{SiO}_2 \frac{\text{Percentage}}{\text{Molecular weight}} = \frac{27.6}{60} = .46.$$

This left a slight excess of silica, which fact has been proved by subsequent analysis to add to the perfect fluxing and fluidity of the slag."

<sup>1</sup> *Jour. Chem. Met. and Min. Soc. of S. Africa*, vol. v. pp. 168-9, 1905.

In calculating the proportion of borax necessary to flux the other oxides,  $\text{Fe}_2\text{O}_3$  is taken as a representative one :

$$\text{Fe}_2\text{O}_3 \frac{\text{Percentage}}{\text{Molecular weight}} = \frac{13}{160} = 0.0875,$$

and the amount is

$$\text{Na}_2\text{B}_4\text{O}_7 \frac{x}{202} = 0.0875 ;$$

$$\therefore x = 17.675 \text{ per cent.}$$

It is, however, inadvisable to adhere to any prescribed rule as to the exact proportion of fluxes required, or to predict the ultimate fineness of the bullion which may result. The latter may vary to a considerable extent with the same method of treatment. One operator may obtain a bullion of 700 fine with extended treatment and complicated refining process. Another operator may handle the identical product, abolish all attempts at refining, and by simple methods obtain a bullion 900 fine. Common-sense modifications will invariably lead to the raising of the efficiency in any process.

The following summary (Table XLV.) will serve to indicate the wide variation of method in the treatment of cyanide precipitate.

**Smelting Operations.**—The fluxed precipitate is either briquetted, or smelted in the condition it leaves the drier or precipitate press. Drying to below 5 per cent. moisture has been found to be unnecessary, except in the instance where benefits arise from the calcining action in the furnace with high temperatures. In any case excessive heating before smelting is liable to lead to dusting and an eventual loss of metal.

To avoid mechanical loss of the fluxed precipitate when being charged into the furnace, the mixture is preferably weighed into small paper bags each holding 4 lb. or so. This method ensures cleanliness, and enables the operator to keep in better touch with the progress of the smelt. Its sole disadvantage lies in the fact that some reducing material is thereby introduced, but the amount is very small.

All operations in the smelting-room should be methodical, and each addition of precipitate to the crucible should be entered in the log-book, together with the time of the addition and the quantity of material added. A furnace can only be worked at maximum efficiency when operations are being conducted in a systematic manner.

In charging briquetted precipitate into a furnace the briquettes are added by means of a suitable scoop, care being taken that the addition of fresh precipitate is made before the mass in the crucible has completely subsided.

In the smelting of cyanide precipitate, the regulation of temperature has much to do with the efficiency of operations. If the temperature is allowed to rise too high during the fusion stage there is violent ebullition, and the capacity of the crucible is considerably reduced. The temperature variation required in an ordinary fire assay may be used as a guide ; and as soon as the whole mass is in a state of quiet fusion the temperature should be regulated so that no undue ebullition occurs. Fresh fluxed precipitate

TABLE XLV.

| Reduction Works<br>(and reference).  | Treatment of<br>Precipitate.  | Fluxes.   | Remarks.  |
|--|---|---|---|
| 1. Nipissing high-grade Mill, Cobalt, Ont., Canada. R. B. Watson, <i>E. and M.J.</i> , Dec. 7, 1912.                   | Retorting to remove mercury. Smelting. Refining with air blast.         | No fluxes.  | Silver ore. Zinc-shaving precipitation. Bullion, 999 fine.  |
| 2. Rio Plata Mill, Chihuahua, Mexico. A. W. Allen.   | Smelting direct.  | Precipitate, 100<br>Borax, 15<br>Sand, 8                                    | Silver ore. Zinc-shaving precipitate. High-grade precipitate (80 per cent.). Bullion, 980-990 fine.   |
| 3. Deloro Mining and Reduction Co.'s Mill, Ontario, Canada. S. F. Kirkpatrick, <i>E. and M.J.</i> , June 28, 1913.     | Smelting direct.  | Precipitate, 100<br>Borax, 16<br>Sod. Nitrate, 8<br>Fluorspar, 4<br>Sand, 5 | Silver ore and speiss. Aluminium dust precipitation. Bullion, 999 fine.                               |
| 4. El Tigre Mill, Sonora, Mexico. F. M'Cann, <i>Cyanide Practice in Mexico</i> .                                       | Smelting direct.  | ...   | Silver ore. Zinc-dust precipitation. Bullion, 850-920 fine.   |
| 5. Homestake Mine, U.S.A. Clark and Sharwood, <i>I.M.M. Bulletin</i> No. 97, 1912.                                     | Acid treatment. Smelting with li-tharge. Cupellation in muffle furnace. | ...   | Gold ore. Zinc-dust precipitation. Bullion, fine.   |
| 6. Westralia Mt. Morgans Mill, W. Australia. A. W. Allen.  | Roasting. Smelting direct.  | Precipitate, 100<br>Borax, 50<br>Sand, 15                                   | Gold ore. Zinc-shaving precipitation. Low-grade precipitate (10-15 per cent.). Bullion, 800-850 fine. |
| 7. Consol. Goldfields Mines, South Africa. Johnson and Caldecott, <i>J.C.M. and M. Soc. of S.A.</i> , vol. iii. p. 46. | Acid treatment. Calcining. Smelting.                                    | Precipitate, 100<br>Borax, 20-35<br>Manganese dioxide, 20-40<br>Sand, 15-40 | Gold ore. Zinc-shaving precipitation. Medium-grade precipitate (35 per cent.). Bullion, 970 fine.     |
| 8. Lonely Reef Mill, Rhodesia. A. W. Allen.  | Acid treatment. Drying. Smelting.                                       | Precipitate, 100<br>Borax, 30<br>Sand, 15<br>Sod. Carbonate, 5              | Gold ore. Zinc-shaving precipitation. Medium-grade precipitate (55 per cent.). Bullion, 900-920 fine. |

can then be added from time to time until the full load has been reached; and, after the mass has quietened down, the temperature can again be raised to ensure a thoroughly fluid pour. Oil fuel offers the best facilities for temperature control.

In some installations the furnaces still consist of the ordinary square type used by assayers, but large enough to take a No. 60 to No. 150 crucible. The top of the furnace is generally about on the same level as the floor for convenience in lifting out the pots. Good oven coke gives the best results as fuel and is easy to manage, but lump coal has often to be used. In some districts charcoal is the only suitable fuel and, where this is the case, a somewhat larger furnace than is used for coke becomes necessary.

Clay liners to the crucibles are generally used in the smelting of acid-treated precipitate. Their life varies with the class of material being smelted, the heat of the furnace, and the quantity of oxidising refiner used.

Recent practice in smelting is almost exclusively confined to the use of the tilting furnace. The Western Australian type is fired with coke, with or without a small addition of charcoal. The most modern adaptation of this furnace is seen in the Morgan oil-fired tilting furnace, which is so arranged that the pouring point remains stationary at every stage of the pour. Such a furnace is the acme of cleanliness, and enables a smelt to be carried on with none of the disadvantages apparent in the other systems, and often with a reduction in operating costs.

**Remelting and Refining of Bullion.**—Gold bullion is invariably remelted in small plumbago crucibles in ordinary assay-type furnaces. Refining gold after it has once been melted into buttons is seldom practised. Consideration should be paid to this question while the metal is still in the finely divided condition of a precipitate.

Silver is cast into bars averaging a little over 1000 oz. Gold ingots are generally under this weight.

To avoid unnecessary expense in refining charges, all silver should be sent out from the works as near fine as possible. On the other hand, there is no appreciable saving, other than in transportation costs, to be effected by refining gold bullion above 900 parts per 1000.

**Recovery of Metal from Slag.**—In many cases the slag is crushed and then either amalgamated with mercury in a Berdan pan, or concentrated over a shaking table. In some instances the slag is granulated by pouring into water preparatory to concentration. In either case the concentrated product is treated in the usual way and the residue sold to the smelter.

## CHAPTER XXXIV.

### LEACHING PLANT CONSTRUCTION AND EQUIPMENT.

#### *Section I.*

**Vat Capacity in Cyanide Plants.**—When the nature of the material to be treated and the method to be adopted have been determined by preliminary tests or by previous work in the same locality, the total vat capacity required in the plant for treatment is easily estimated by multiplying the daily tonnage, expressed in cubic feet, by the total number of days occupied in the treatment of one charge, including, of course, the filling and emptying of the vat. Thus, for instance, in a single-treatment sand plant, if one day is allowed for filling, one for discharging, and five for treatment, the calculation is as follows: for 500 tons a day of material occupying, say, 24 cu. ft. per ton,

$$(a) \quad 500 \times 24 \times 7 = 84,000 \text{ cu. ft.}$$

When collecting vats are to be used, then, on the basis of one day to fill, one to drain, and one to empty, the capacity required at 20 cu. ft. per ton for direct filling is as follows:

$$(b) \quad 500 \times 20 \times 3 = 30,000 \text{ cu. ft.}$$

For a double-treatment plant, example (a) would be suitable for the second series of vats. For the collection and first-treatment vats, allowing two days for filling and discharging and two for preliminary treatment:

$$(c) \quad 500 \times 20 \times 4 = 40,000 \text{ cu. ft.}$$

This will be sufficient if the plant is so arranged that the contents of any collecting vat can be transferred to any vat of the second series, but with double-tier plants the same time must be allowed in reckoning the capacity of each set, and the only difference in the respective sizes is due to the difference in the volume occupied by the material in the upper and lower vats.

In the case of slime plants, it is necessary not only to allow for the volume of the slime, but also for the solution used with it. Thus, if a 4 to 1 pulp is to be used in the agitation and settling vats, the total capacity is found in the following way: for, say, 100 tons of dry slime daily, with a total treatment time of 144 hours:

$$100 \text{ tons slime at } 12.3 \text{ cu. ft.} = 1,230 \text{ cu. ft.}$$

$$400 \text{ tons water at } 32 \text{ cu. ft.} = 12,800 \text{ ,,}$$

$$\text{Total volume treated daily,} = 14,030 \text{ cu. ft.}$$

which, multiplied by 6 days, is equal to 84,180 cu. ft.

The above simple calculations are sometimes useful in rough estimations, for the purpose of comparison of plants for different methods of treatment, and also for checking the results obtained from the considerations mentioned hereafter.

For solution vats, the total storage capacity usually provided in sand or slime plants is between one-fifth and one-tenth of the total treatment-vat capacity.

**Number and Size of Vats.**—As the time of filling and emptying is necessarily included in the above figures, it follows that the total capacity will to some extent finally depend on the number and size of the vats.

For this and many other reasons in designing a plant, instead of calculating the total capacity, it is better to select first of all the size of the vat, and then to decide upon the shape, i.e. the relation between diameter and depth, which is most suitable for the particular material in each case.

When treating slimes in vats fitted with revolving arms, there are self-evident mechanical reasons for not making the vats very large. But in plants where the entire treatment consists of a series of transfers from one vat to another by centrifugal pumps, there is practically no limit to the size allowable for the tanks. Each vat should, however, be large enough to give a clear water overflow when receiving the whole slime product from the mill, and need not be larger. Suppose, as an illustration of this point, that the slime product from a 40-stamp mill will give a clear overflow from a 24-ft. vat which fills in 6 hours, and that the total treatment under these conditions takes 60 hours. In order that the time required for settlement may be constant, it may be assumed that the diameter only is altered, so that the vat will receive a charge equal to 24 hours' supply from the mill. Then the total treatment time will be increased to 78 hours or more, and the total capacity required will be increased in the same proportion, that is to say, by 30 per cent. If, on the other hand, the depth is increased, the settlement time will also be lengthened, and the necessary capacity still more seriously increased.

**Most Economical Shape of Vat.**—When only the cost of the vat itself is considered, it has been shown mathematically, for a vat of uniform material and thickness throughout, that the most economical shape is obtained when depth equals radius. But the cost of foundations has also to be taken into account, and as this item also increases with the diameter, it follows that the depth should be greater than the radius, to secure the greatest economy in construction. Suppose, as a simple example, that the cost of foundations per square foot covered is exactly equal to that of the vat bottom per square foot, then the cheapest vat of uniform thickness will be one in which the depth is equal to the diameter. But in most practical cases for treatment vats, and certainly where steel tanks are elevated on columns and girders, constructional economy demands a still greater depth. However, it is only for vats of comparatively small size that uniformity of thickness can obtain. When the size is so large that the internal pressure becomes the determining factor as



regards the strength of the sides, these must increase in thickness as the diameter increases, whereas the bottom need not increase in thickness after a certain limit, as it is supported equally well over its whole area.

With steel tanks, for instance, the most economical shape can be determined as follows :

Let  $D$  = diameter and  $d$  = depth, both in feet ;  
and  $C$  = contents in cubic feet.

$$\text{Then } C = \frac{\pi D^2 d}{4}.$$

The quantity of material in the bottom is proportional to  $\frac{a\pi l^2}{4}$ , where  $a$  is the thickness in inches.

The thickness of the sides is equal to  $bDd$ , where  $b$  is a coefficient depending upon the stress allowed on the steel. Therefore the total material in the sides is proportional to  $b\pi D^2 d^2$ . Then the relative cost of the vat is given by

$$\begin{aligned} \frac{\text{Material}}{\text{Contents}} &= \frac{\frac{a\pi D^2}{4} + b\pi l^2 d^2}{\frac{\pi D^2 d}{4}} \\ &= \frac{a + 4bd^2}{d} \quad \dots \dots \dots (1) \end{aligned}$$

As  $D$  has disappeared, it shows that the diameter has no effect upon the economy.

The minimum value of this expression is obtained when

$$d = \frac{1}{2} \sqrt{\frac{a}{b}} \quad \dots \dots \dots (2)$$

As an example take  $\frac{5}{16}$  in. as the standard thickness for the bottom of large steel tanks, and  $b = 0.00058$ , which is the coefficient given in equation (4).

$$\text{Then } d = \frac{1}{2} \sqrt{\frac{.3125}{.00058}}$$

which gives 11.6 ft. as the most economical depth under these conditions.

But as in the former case, so in this, the cost of foundations will require a still greater depth for the greatest economy. It is evident therefore that, as a rule, the most economical shape cannot be used in practice, as other considerations, such as effective percolation and convenience in working, have more importance than a small saving in first cost. Solution storage vats can, however, generally be made much deeper than leaching vats; and as their foundations are seldom costly, the best depth can often be adopted, this being usually between one-half and three-quarters of the diameter. In all cases it is useful, when designing a plant, to remember in which direction economy lies.

**Pressure in Vats.**—Small vats are usually designed according to precedent, and calculations are seldom necessary, but for large sizes the stresses should be carefully determined and allowed for.

When a vat is filled with sand and water the whole mass forms what is practically a quicksand, and, so far as its pressure is concerned, it may be considered to act as a fluid, whose weight is equal to that of the sand and water together. The two extreme cases may be taken to find the limits of this weight for quartz sand.

|                                 |                                 |
|---------------------------------|---------------------------------|
| Mixed sand, 1 cu. ft. = 100 lb. | Classified sand, 1 ft. = 80 lb. |
| Water, 0.392 cu. ft., say 24 „  | Water, 0.52 cu. ft., say 33 „   |
| Total, 124 lb.                  | Total, 113 lb.                  |

In the following examples 120 lb. will be taken as the weight per cubic foot of the fluid mass. For pyritic concentrates a higher figure should, of course, be used, which can be found in the same way.

The pressure in pounds per square foot upon the side of the vat at any depth will be found by multiplying the depth in feet by 120. The figure so obtained multiplied by the diameter in feet will give the total pressure tending to burst the vat at the depth selected. This, of course, is zero at the top, and reaches its maximum at the bottom.

**Design of Timber Vats.**—In a wooden vat there is an additional stress on the hoops, due to the swelling of the staves and planks, but in the absence of exact data as to the amount of stress resulting from this cause, it is allowed for in practice by giving a little extra strength to the hoops generally, and by adding one hoop at the bottom, in order to resist the swelling of the floor.

One convenient method for designing a wooden vat is to calculate the stresses separately for each 1 ft. of depth, and so determine the necessary amount of iron required in the hoops at each part, because by this means the approximate spacing of the hoops is found, as well as their number and size. As only a few sizes of round iron are usually employed in vat construction, it is possible to assign a value for the safe load on each size. Taking 11,200 lb. per square inch, and allowing for cutting threads at the ends, then for round iron:

|  |                            |
|--|----------------------------|
| $\frac{3}{4}$ in. diameter, . . .                      | the safe load is 3,405 lb. |
| $\frac{7}{8}$ „ „ . . .                                | „ „ „ 4,726 „              |
| 1 „ „ . . .  | „ „ „ 6,205 „              |
| $1\frac{1}{8}$ „ „ . . .                               | „ „ „ 7,806 „              |
| $1\frac{1}{8}$ with $1\frac{1}{2}$ screwed ends, . . . | „ „ „ 11,133 „             |

Larger sizes than these are not advisable, as there is much difficulty in bringing them well up to every stave.

As an example, the necessary calculations will be made for a vat 40 ft. in diameter and 8 ft. deep inside. When working in this way it is well to draw a vertical line, such as AB in Fig. 51, to any convenient scale to represent the side of the vat, with the number of feet marked and numbered from the top. Then the position of each hoop can be tentatively marked off on this line, and afterwards a final slight adjustment of positions can be made if desirable.

Beginning at the bottom, the pressure at 8 ft., tending to burst the vat, is

$$120 \times 8 \times 40 = 38,400 \text{ lb., and at 7 ft. it is } 33,600 \text{ lb.}$$



It is well known that the total tensile stress in all the hoops is then equal to the area of the triangle ABF, i.e. to  $BF \times \frac{AB}{2}$ . This total stress divided by the safe load on each hoop will give the number of hoops required. Further, if the triangle ABF is divided by horizontal lines,  $i_1 k_1$ ,  $i_2 k_2$ , etc., into as many equal areas as the number of hoops, then the intersections of these lines with the line AB will divide the latter in such a manner that each portion  $Bi_1$ ,  $i_1 i_2$ , etc., can be safely supported by one hoop. The construction necessary for this is the following: Bisect AB and describe a semicircle on it. Then divide AB into the required number of equal parts as at  $g_1 g_2 g_3 g_4$ .

From each of these points draw horizontal lines cutting the semicircle in  $h_1 h_2$ , etc. Then with centre A and the respective radii  $Ah_1$ ,  $Ah_2$ , etc., draw circular arcs cutting AB in  $i_1 i_2$ , etc., and through these points draw horizontal lines  $i_1 k_1$ ,  $i_2 k_2$ , etc., which will divide the triangle ABF into equal areas as required.

As each of these areas, for instance  $Bi_1 k_1 F$ , represents the pressure sustained by one hoop, it follows that a horizontal line through the centre of gravity of  $Bi_1 k_1 F$  will represent the resultant pressure, and the point where it cuts the line AB will be the correct position for the hoop. But for the purpose under consideration it is sufficient, and for some reasons better, to place the hoops in the following manner, so that, considering each section separately, the pressures above and below the hoop are equal. It is obvious that this can be done by dividing AB in the first operation into twice as many parts as there are hoops, by means of the intermediate points  $n_5 n_4 n_3$ , etc., and by the same construction transferring these points to  $l_5 l_4$ , etc., on the semicircle, and back to the line AB at  $m_5 m_4$ , etc., which will be the required positions for the hoops. In this way the trouble of finding the centre of gravity of each separate area is avoided, and the extra strength is provided to withstand the pressure due to the swelling of the staves.

Except for the purpose of clearly explaining the principles underlying this method, the drawing of the triangle and its subdivisions is not actually necessary. In the next diagram, Fig. 54, which is drawn to scale for a vat 30 ft. in diameter and 8 ft. deep, all unnecessary lines are omitted. The bursting pressure at the bottom in this case is 28,800 lb., and therefore the total tensile stress in all the hoops is  $\frac{28,800}{2} \times \frac{8}{2} = 57,600$  lb. If hoops of

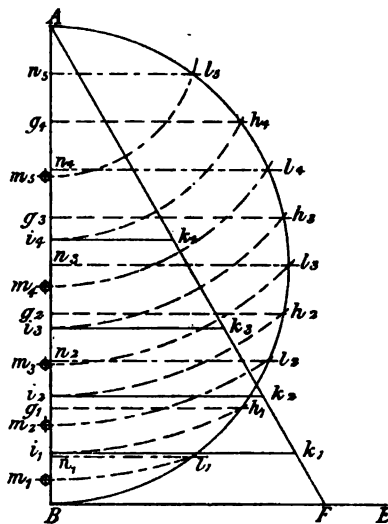


FIG. 53.—Graphic Method of Spacing Hoops.

$1\frac{1}{8}$ -in. round iron with  $1\frac{1}{4}$ -in. screwed ends are used, then 5.2 hoops will be required. The line AB is therefore divided into 10.4 parts and the positions  $m_1 \dots m_6$  of the hoops are found as before.

But by scaling from the diagram it is found that the distance between  $m_4$  and  $m_6$  is 19 in., so it is better to use two hoops of smaller size in place of  $m_6$ . In Fig. 54 the upper part of the stave is dealt with separately by dividing it into four equal parts, and thus finding the positions,  $m_6, m_7$ , for two hoops of 1 inch diameter throughout. In addition to these, it is necessary to

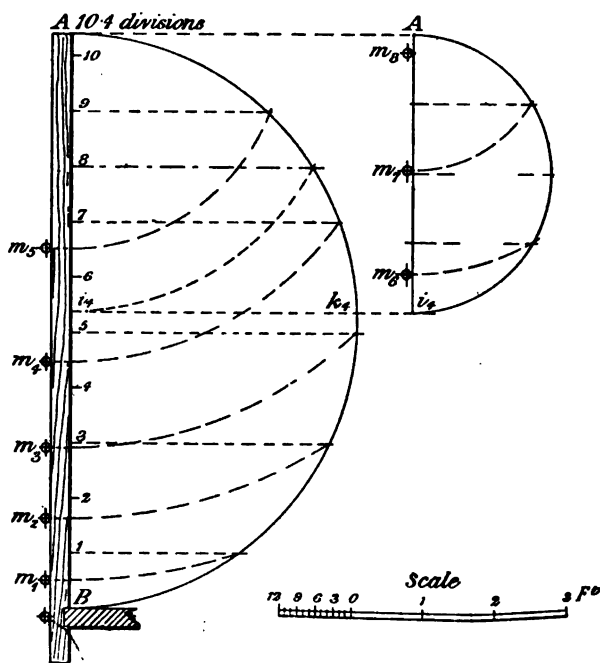


FIG. 54.—Graphic Method of Spacing Hoops on a 30-foot Vat.

put an extra hoop about 3 in. from the top. Theoretically this could be very thin, but in practice it is generally made the same size as those immediately below it, to avoid multiplicity of sizes. An extra hoop around the bottom completes the number.

As regards the floor of the vat, it is only necessary to see that the material of which it is composed is strong enough to carry this load over the spaces between the joists or other supports, but this is a matter that is usually considered in arranging such supports rather than in the design of the vat itself.

**Construction of Timber Vats.**—Vats are usually made of either red deal, Oregon pine, Californian redwood, or Kauri pine.

For small clean-up vats up to 6 ft. diameter, staves 4 in. wide by  $1\frac{1}{2}$  in. thick are used, with  $\frac{1}{2}$ -in. check on to bottom and 3-in. chine.

For vats up to 12 ft. diameter the staves may be  $4\frac{1}{2}$  in. wide by 2 in. thick, with  $\frac{5}{8}$ -in. check and 4-in. chine. Up to 25 ft., staves 6 in. wide, and for larger vats 9 in. wide, are usual, the thickness being 3 in., check  $\frac{3}{4}$  in., and chine 6 inches.

The staves are planed to the proper bevel on both edges by hand or machine; if by the latter, they should be hand-dressed afterwards. In first-class work the staves are also planed on both faces and thickened. The planks forming the floor of the vat are usually of the same thickness as the staves, but somewhat wider, being 6 in. for small sizes, 9 in. for medium sizes, and 12 in. for vats more than 25 ft. in diameter. They should be planed in both faces and thickened, planed and hand-dressed truly square on the

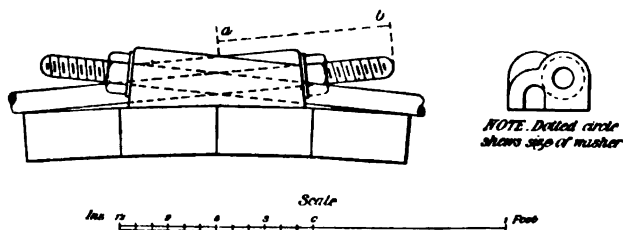


FIG. 55.—Hoop Casting.

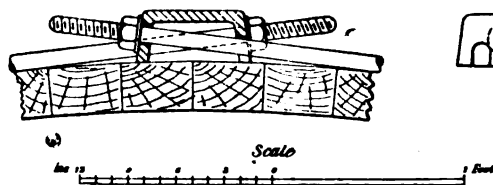


FIG. 56.—Hoop Casting.

edges, which are generally grooved  $\frac{3}{8}$  in. wide and  $\frac{1}{2}$  in. deep, to receive clear pine fillets,  $\frac{3}{8}$  in. thick and 1 in. wide.

The hoops are usually made of round iron for small sizes, because they can be screwed at the ends, whereas with flat bar, separate screwed ends have to be welded on. For large sizes it is better to weld on screwed ends whose section inside the thread is equal to that of the bar. In some districts the saving of material in this way is greater than the cost of welding, but the main object is to avoid any unnecessary metal in the hoops, because any excess is worse than useless, as it increases the difficulty of fitting them closely around the staves. So for large hoops some prefer to use flat, bar iron.

The hoops are made in several sections, which are connected by castings of the form shown in Fig. 55, or in Fig. 56. It is clear that each section must overlap the next one by an amount equal to  $ab$  (see Fig. 55) at each end. If the overlap be made too long, it means expense of extra screwing at each end; but when it is too short, there is trouble in getting on the last nut of each hoop. A good practical length is obtained by making  $ab$  9 in. for 20-ft. vats and, say, 11 in. for 40-ft. ones. The thickness of metal in

castings should be about  $\frac{1}{4}$  in. less than the outside diameter of screwed ends of the rods.

The check in the stave to fit the bottom is sometimes made with parallel sides, as in Fig. 57, in which case the bottom is left flat on the under side, and is only slightly bevelled on the upper side. Some makers prefer a check with inclined sides: in this case the upper and lower sides of the bottom are bevelled to the same angle as in Fig. 57. Either method gives satisfactory results, provided that the workmanship is good.

Timber vats are erected in the following manner: The joists are first placed in position and properly bedded, so that their upper edges are all truly level and in the same horizontal plane. The bottom planks are then laid down, the fillets fitted in the grooves, and the whole tightly cramped up or wedged. While cramped, cross strips are temporarily screwed down to the planks to hold the latter in place. The cramps are then removed, and a circle struck out whose diameter is from 1 to  $1\frac{1}{2}$  in. larger than the inside dimension of the vat, according to the depth of the check in the staves. The bottom is then cut to this circle and bevelled around the edge to fit the checks.

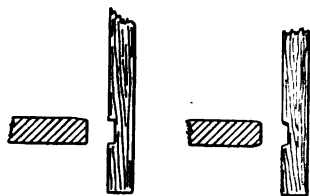


FIG. 57.—Stave Checks.

In putting on the staves it is important that each stave, as it is put in place, should be driven on to the bottom as far as it will go, and also close to the previous stave. For if the first point is not attended to, the circle of staves when completed will be too large, and it will then be impossible to close the staves tightly on to the bottom. If the second matter is neglected, so that the staves when placed are as close as possible to the bottom, but not close to each other, it will then be impossible to close up the joints between the staves, except, of course, near the top, where they can yield to the pressure of the hoops. A little experience and care soon enables a skilled joiner to adjust matters, so that, when the hoops are screwed up, the staves will close tightly against each other and, at the same time, against the bottom planks. If the completed vat leaks between the staves, a thin extra stave is needed, but if it leaks round the edge of the bottom, one of the staves must be made narrower.

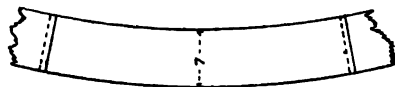


FIG. 58.—Plank for bottom of Launder.

When all the staves are placed, a few hoops are put on and partly tightened up, after which the remaining hoops are placed in position and all are gradually adjusted.

Spanners up to 30 in. long are used; and each hoop, while it is being drawn up, is well hammered, to bring it close to each stave. The joints in the several hoops should not be placed one over another, but should be placed 'en échelon,' as shown in the various drawings of vats.

Some engineers allow the abutting edges of the staves and floor planks, and also the tongues of the latter, to be smeared with a very thin layer of white lead in oil, whereas others prefer to rely only on the dressing of the timber for a tight joint. In Australia, the grooves and tongues in the floor are generally not used.

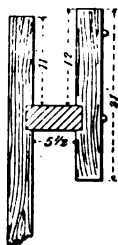
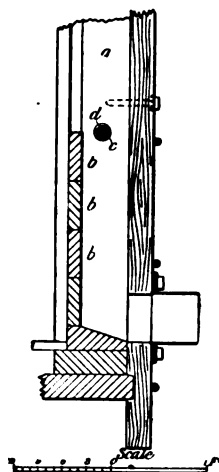
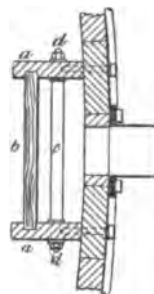


FIG. 59.—  
Cross-section  
of Launder.

An excellent method of making the joints thoroughly watertight, which may be adopted when very rich solutions are to be dealt with, is described by J. Mactear<sup>1</sup> as follows: The planks to be used for staves are passed edgewise between a pair of rollers provided with a central projecting rib, under a considerable pressure obtained by weights and levers. These form a central groove  $\frac{1}{2}$  in. to  $\frac{3}{4}$  in. in breadth along each edge of the plank by compressing the fibre of the wood to a depth of  $\frac{1}{2}$  in. When the plank afterwards passes through the planing machine, the whole of the wood projecting beyond this groove is planed to the proper bevel, so that after the tank is built and wetted, the compressed fibre expands and forms a tight joint. This method, of course, involves the waste of 1 in. of material from each stave, which is, however, not a serious matter.

**Timber Collecting Vats.**—In these, a circular launder is formed round the outside of the vat at the top in the following manner: Each stave, before erection, is checked about  $\frac{1}{2}$  in. deep on the outside, so that these checks form a continuous groove round the vat about 5 to 12 in. from the top. The bottom of the launder is formed from 3-in. planks cut to the shape shown in Fig. 58, which are fitted into the groove just mentioned. Short staves are driven on the outside edge of these curved pieces, and drawn together by a couple of light hoops, say of  $\frac{3}{4}$ - or 1-in. round iron, the result being shown in Fig. 59, which is a cross-section through the trough so formed.

A slat gate for a wooden vat is shown in Fig. 60. Two uprights *a a*, fixed inside the staves, have grooves to receive the slats *b b*. Short pieces of pipe *c c*, with washers threaded on bolts *d d*, serve to keep the uprights from warping. The outflow orifice may be fitted with any suitable form of valve, if any cyanide treatment is to be carried on in the vat. When used for collection only, a spout cut from a 6-in. pipe can be attached to the vat by an ordinary pipe flange as shown. Suitable widths for these slat gates are—for 20-ft. vat, 1 ft. 6 in.; for 25-ft., 1 ft. 9 in.; and for 30-ft. vat, 2 feet.



FIGS. 60, 61.—Slat Gate.  
Plan and Section.

<sup>1</sup> *Proc. Inst. C.E.*, 1894-5, vol. cxxi. part iii. p. 22.



**Quantities in Timber Vats.**—By the use of either of the following formulæ the quantity of timber in a vat of any size from 10 to 40 ft. in diameter can be quickly found. The waste of timber in cutting the circular bottoms is allowed for, and the results are correct to about  $1\frac{1}{2}$  per cent. of the total.

Let  $D$  = inside diameter of vat in feet.

$L$  = length of stave in feet.

$F$  = number of feet of 9 in.  $\times$  3 in. deals.

$C$  = quantity of timber in cubic feet.

$S$  = sq. ft. of timber reduced to 1 in. thick.

Then,

$$F = aDL + bD^2.$$

$$C = cDL + dD^2.$$

$$S = eDL + fD^2.$$

The constants of the first terms are—

$$a = 4.35$$

$$c = 0.816$$

$$e = 9.787.$$

The coefficients of the second term vary slightly with the diameter and are therefore tabulated below.

TABLE XLVI.—*Values of Coefficients b, d, and f.*

| Diameter of Vat in feet. | $b$   | $d$  | $f$   |
|--------------------------|-------|------|-------|
| 10                       | 1.210 | .227 | 2.724 |
| 15                       | 1.194 | .224 | 2.688 |
| 20                       | 1.178 | .221 | 2.652 |
| 25                       | 1.162 | .218 | 2.616 |
| 30                       | 1.147 | .216 | 2.592 |
| 35                       | 1.145 | .215 | 2.580 |
| 40                       | 1.143 | .214 | 2.568 |

The same formulæ can be used for collecting vats by adding 2 ft. 6 in. to the actual length of the stave, to allow for the timber in the peripheral overflow trough.

For the quantity of iron in the hoops there is no simple or exact formulæ, but the one given below gives figures within 10 per cent. of the correct weight for hoops arranged according to the methods previously described. Thus, if  $W$  = total weight of iron in pounds, then

$$W = (D + 31)^2 (L + 4)^2 \times .004.$$

This formula appears complicated for the purpose, but when it is used with a table of squares, such as is found in engineering pocket-books, the amount of work involved is much less than that of taking out the weights in detail, unless working drawings are available. The weight of the cast-iron connections can be taken at one-half to two-thirds of the weight of the hoops.

**Timber in Joists.**—For deep vats with discharging tunnels beneath them, 9  $\times$  6-in. joists are generally used, spaced at 2 ft. 3 in. centres, and for

shallow vats, 9 × 3 in. joists at 2-ft. centres are sufficient. The first case, including herring-bone struts, requires nearly 1 ft. of 9 × 3-in. material per square foot of tank bottom, and the latter case, a little more than half this amount. Therefore, to obtain the quantity of timber in the joists, it is only necessary to multiply the area of the bottom by some number between 1 and 0·5, depending upon the depth of the vat and the maximum width of opening spanned by the joists. The latter is usually not more than 6 ft. or less than 5 ft., and the following formulæ, with the same notation as before, will, in average cases, give an approximately correct result :

$$F = 0\cdot0436 D^2(L + 4).$$

$$C = 0\cdot0082 D^2(L + 4).$$

$$S = 0\cdot0981 D^2(L + 4).$$

The above empirical formulæ, and others of the same kind which will be given later, are not intended for use in close estimating when drawings have been made, but they will be found useful in making rough estimates of cost, and for obtaining the relative costs of vats of various sizes.

## CHAPTER XXXV.

### LEACHING PLANT CONSTRUCTION AND EQUIPMENT.

#### Section II.

**Strength of Steel Vats.**—In designing large steel vats, the quickest method is to use formulæ, such as those given below, to find at once the thickness of metal required at any depth. Generally, it is only necessary to make the calculation for the full depth, and to use the same thickness throughout.

Let  $d$  = depth in feet from surface.

$w$  = weight of one cubic foot of the contents of the vat. Say 120 lb.  
for average sand.

$p$  = pressure per square foot at depth  $d$ .

$D$  = diameter of vat.

$P$  = total bursting pressure in pounds on any horizontal slice of the vat,  
one foot in height whose mean depth is  $d$ .

$t$  = thickness of metal in side of vat.

$k$  = safe tensile stress in pounds per square inch.

*Note.*—For steel plates with single-riveted lap joints, or butts with single covers,  $k = 8620$  lb.

Then

$$p = dw, \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

$$\therefore P = Ddw \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

The section of steel in square inches for each foot of depth =  $12 \times 2t$ , and the safe stress is therefore  $24 kt$ , and this must equal  $Ddw$ .

$$\text{Therefore } t = \frac{Ddw}{24k} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

This is a general formula suitable for all cases, but taking the values for  $w$  and  $k$  given above, then for a sand vat,

$$t = \frac{Dd}{1724} = 0.00058Dd \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

As an example, with a vat 30 ft. in diameter and 10 ft. deep: By equation (4)  $t = 0.174$ , or say  $\frac{3}{16}$  in. But, for constructional reasons, a thinner plate than this should never be used, even for small vats. Therefore, for vats of smaller diameter, no calculations as to stress need be made, because the thickness required for rigidity and to ensure watertight joints is always more than sufficient to sustain safely the internal pressure in these cases.

For tanks with spherical bottoms, supported round their outer edge only, the required thickness can be found for the bottom plates by substituting  $r$  = radius of sphere, in place of  $D$  in the formulæ given above. Thus (4) becomes

$$t_s = 0.00058rd.$$

As before, with a tank 30 ft. diameter and 10 ft. deep at the sides, but in this case with a spherical bottom curved to a 30-ft. radius, the depth at the centre will be 14 ft. and the thickness must be 0.224, or say  $\frac{1}{4}$  in.

The following dimensions represent ordinary practice as regards thickness of plates :

Up to 25 ft. diameter and 10 ft. deep,  $\frac{3}{8}$  in. throughout. From 25 to 30 ft. diameter, if made in two rings, the upper ring may be  $\frac{3}{8}$  in., the lower ring and bottom  $\frac{1}{2}$  in., from 30 to 35 ft.  $\frac{1}{2}$  in. throughout, from 35 to 40 ft.  $\frac{1}{2}$  in. sides and  $\frac{5}{8}$  in. bottom.

For single-riveted lap joints the following sizes are suitable :

|                               |                           |                           |                         |
|-------------------------------|---------------------------|---------------------------|-------------------------|
| For $\frac{3}{8}$ -in. plates | $\frac{7}{8}$ -in. rivets | $1\frac{1}{2}$ -in. pitch | $1\frac{1}{2}$ -in. lap |
| " $\frac{1}{2}$ -in. "        | $\frac{7}{8}$ -in. "      | $1\frac{3}{4}$ -in. "     | $1\frac{1}{2}$ -in. "   |
| " $\frac{5}{8}$ -in. "        | $1\frac{1}{8}$ in. "      | $1\frac{3}{4}$ -in. "     | $2\frac{1}{2}$ -in. "   |
| " $\frac{3}{4}$ -in. "        | $\frac{1}{2}$ -in. "      | $1\frac{1}{2}$ -in. "     | $2\frac{1}{2}$ -in. "   |

Angle-iron rings at top and bottom vary from  $2\frac{1}{2} \times 2\frac{1}{2} \times \frac{1}{4}$  in. for 20-ft. vats, up to  $3\frac{1}{2} \times 3\frac{1}{2} \times \frac{5}{8}$  in. for 40 ft. diameter, and the top angle may be  $\frac{1}{2}$  in. narrower and  $\frac{1}{8}$  in. thinner than the bottom one. The pitch of rivets for the angle irons should be from four to five times that adopted for the seams.

When ordering vats for export ready for erection, the above particulars, in addition to the overall dimensions and position of discharge doors and other attachments, are sufficient. The following information as to plates was kindly furnished by P. & W. Maclellan, Ltd., of Glasgow, in answer to an inquiry :

Limiting sizes, to be used without incurring extras, 25 ft. by 5 ft. for plates  $\frac{1}{4}$  in. thick, and 25 ft. by 6 ft. for plates  $\frac{5}{8}$  in. thick.

As regards the arrangement of plates, the same firm prefers to place the side plates vertically, each being the full depth of the tank, rather than in horizontal rings with lap joints. This does not apply to deep vats of the Pachuca type.

When, however, tanks have to be built at or near the mines, there are two reasons why the latter arrangement is usually adopted. First, because only small plates can be obtained ; and secondly, because the vertical arrangement requires bending rolls, whose width is equal to the depth of the tank, and such heavy machinery is not usually available in mining districts. The best joints can be obtained, under these circumstances, by using butt joints with cover plates for the vertical seams and lap joints for the horizontal seams (see Fig. 62). Also, in the bottom, the ends of the plates may with advantage be butted and covered. The only objection to this is the cost

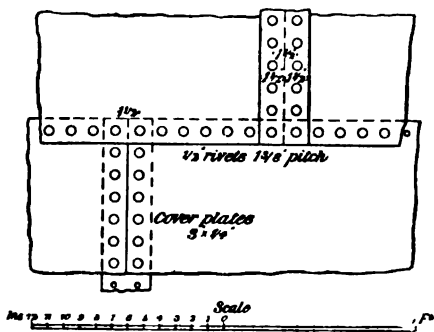


FIG. 62.—Tank Joints.

of cover plates and extra riveting. Butt joints are, however, seldom used for tanks.

In laying out the plan for vat bottoms with small plates, the position of the discharge doors should be first determined, and the plates so arranged that the lines of rivets are clear of the doors. The reduction of waste to a minimum in cutting the circle should also be carefully considered. In Fig. 63 a suitable arrangement of small plates for a vat 30 ft. in diameter is illustrated. The projecting plates on the right-hand side of the figure are made of such lengths that the halves marked A, B, and C will be large enough to provide for the spaces *a*, *b*, and *c* on the left-hand side, so that the quantity cut to waste is small. But, although it is quite possible to make satisfactory tanks with small plates, it is clear that, from every point of view, except as to time of delivery, the advantage lies with tanks constructed of large plates in well-equipped works.

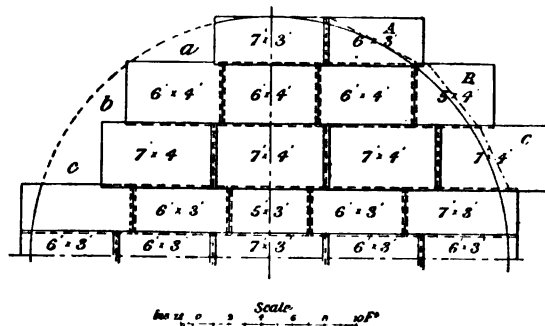


FIG. 63.—Arrangement of Bottom Plates.

When the plates are marked for the rivet holes, they should also be numbered for erection, so that there will be no difficulty in assembling them in their proper relative positions.

For erection, the following amount of labour is required in addition to a foreman—one riveter, one heater, and one holder-up, and two or three labourers. A gang of this number should drive from forty to fifty rivets per hour of actual riveting, the number in any particular case depending slightly upon the size of rivet, but chiefly upon the facilities provided for heating and supplying the rivets. In bolting the bottom plates together, preparatory to riveting, they must be packed sufficiently to allow the holder-up to get underneath. When the riveting of the bottom plates and angle iron is completed, the whole bottom is lowered gradually and evenly on to its permanent seat. The side plates are then bolted to the angle ring and to each other temporarily, and the riveting of the side proceeded with. When the sides consist of two or more horizontal belts, each of these is finished separately.

With plates of  $\frac{1}{4}$  in. or more in thickness, the joints can then be caulked with a tool which has been ground to a cutting angle of about  $70^\circ$ . With thin plates which cannot be caulked in a satisfactory way, the joints are



not only for estimating the cost of the vats, but also in designing the supporting structure, especially when columns and girders are used.

TABLE XLVII.—*Weight of Material in Steel Vats, including waste in cutting, also Angle Irons, Rivets, and Laps.*

| Inside Depth<br>in feet. | Diameter in feet. |        |        |        |        |
|--------------------------|-------------------|--------|--------|--------|--------|
|                          | 20                | 25     | 30     | 35     | 40     |
| 6                        | 6,890             | 9,020  | 15,180 | 19,720 | 29,410 |
| 7                        | 6,920             | 9,660  | 16,300 | 20,990 | 30,780 |
| 8                        | 7,440             | 10,290 | 17,370 | 22,240 | 32,160 |
| 9                        | 7,960             | 10,910 | 18,380 | 23,470 | 33,550 |
| 10                       | 8,470             | 11,550 | 19,390 | 24,690 | 34,950 |

The figures for 30-ft. diameter vats appear, at first sight, to be abnormal, relatively to those in the other columns. This is accounted for, however, by the change from  $\frac{3}{8}$ - to  $\frac{1}{2}$ -in. plates at that diameter.

Tables XLVIII. to L., giving the quantities for steel tanks, have been kindly contributed by J. Frier, manager of Butters' Copala Syndicate, Mexico. In South African practice the angles are generally heavier for the larger-sized tanks than those allowed for in the tables; and, further, for tanks of 9 ft. deep and over, a ring of tee section is usually added at half the height; nevertheless, in a 40-ft. tank, 10 ft. deep, these differences only amount to about 2 per cent.

**Masonry Vats.**—In a few cases, cyanide vats have been formed by excavations lined with brickwork set in cement; and in hard, solid ground these have proved satisfactory from an engineering point of view. But they should never be constructed on sites where there is any chance of settlement, for, although this may be sufficiently overcome as regards the strength and stability of the tank considered merely as a mechanical structure, it may, nevertheless, cause fine cracks in the cement lining, which result in leakage of solution that cannot be recovered.

Brick tanks have also been constructed on the surface of the ground, but this construction is seldom adopted, so that it is not necessary to give any detailed rules for their design. The walls must be built in the same way as low masonry dams, but for a fluid weighing 120 lb. instead of 62.4 lb. per cubic foot. With this difference kept in mind, the ordinary rules given in engineering books of reference will apply.

## FOUNDATIONS AND SUPPORTS FOR VATS.

**Timber Framing.**—Vats should be carried on timber supports only in temporary plants, or in cases where timber is so cheap that the saving in first cost will outweigh the disadvantages of this construction. Fig. 65 shows a timber lining for a tunnel excavated in the ground, and the cheapest method of supporting a leaching vat above it. The sleepers should be solidly bedded, well beaten down, and allowed to settle for a few days, under a considerable weight if possible, before the joists are permanently fixed upon them. Each joist should be carefully scribed and fitted to the sleepers so that all are flush and level. The framing, lagging, and sleepers should be tarred before putting in place; and if the work is well done, this construction may be

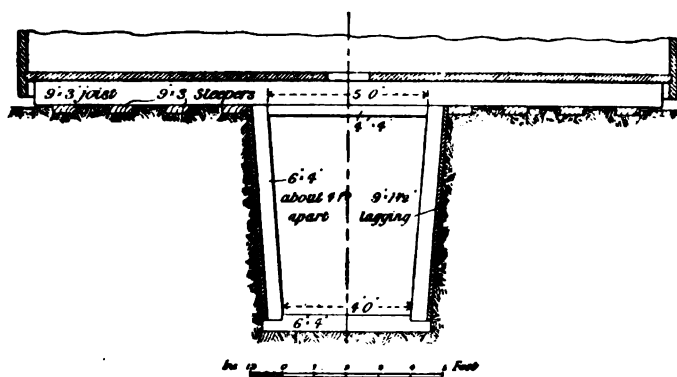


FIG. 65.—Timber-lined Tunnel.

relied upon to last for several years. The frames should be placed at such distances apart that each frame comes under a joist, to which the top distance piece can be nailed. Otherwise the latter would need a much greater scantling.

The upper part of Fig. 66 (p. 280) represents a part plan of a timber frame for carrying a 24-ft. vat above the ground, and the lower part of a section through the centre of the same. In designing a support of this kind, it is necessary to lay out the circle of the vat, and the position of the discharge doors and joists. The frames can then be arranged to leave a clear track under the doors, and also to support the joists to the best advantage. Thus, for instance, particular care should be taken that overhanging ends of joists, such as that shown at B, shall be as few and as short as possible. A sufficiency of diagonal struts in both directions, namely, in the separate frames and at right angles to them, is also of the utmost importance. Framing of this description should never be bedded on the ground, but upon good stone sleeper walls built in lime or cement, or upon well-driven piles.



TABLE XLVIII.—*Approximate Weight in Pounds of Material in Sides of Steel Vats, including two Angle-Iron Rings  $2\frac{1}{4}'' \times 2\frac{1}{4}'' \times \frac{3}{8}''$ , Rivets and Allowance for Laps.*

| Depths of Sides.<br>ft. ins. |               | DIAMETERS IN FEET. |      |      |      |      |      |      |      |      |      |      |       |       |       |       |       | 45    |
|------------------------------|---------------|--------------------|------|------|------|------|------|------|------|------|------|------|-------|-------|-------|-------|-------|-------|
|                              |               | 12                 | 14   | 15   | 16   | 18   | 20   | 22   | 24   | 25   | 26   | 28   | 30    | 32    | 34    | 35    | 36    |       |
| Thickness of Plates.<br>in.  |               |                    |      |      |      |      |      |      |      |      |      |      |       |       |       |       |       |       |
| 4                            | 0             | 1200               | 1400 | 1500 | 1600 | 1800 | 2000 | 2200 | 2400 | 2500 | 2600 | 2800 | 3000  | 3200  | 3400  | 3500  | 3600  | 4500  |
|                              | $\frac{1}{8}$ | 1700               | 1900 | 2100 | 2200 | 2400 | 2700 | 3000 | 3300 | 3400 | 3500 | 3800 | 4100  | 4400  | 4600  | 4800  | 4900  | 6100  |
|                              | $\frac{1}{4}$ | 2000               | 2400 | 2500 | 2700 | 3100 | 3400 | 3700 | 4000 | 4200 | 4400 | 4800 | 5100  | 5400  | 5600  | 6000  | 6100  | 7600  |
|                              | $\frac{3}{8}$ | 2500               | 2900 | 3100 | 3300 | 3700 | 4100 | 4500 | 4900 | 5100 | 5400 | 5700 | 6100  | 6600  | 7000  | 7200  | 7400  | 9200  |
| 4                            | 6             | 1300               | 1500 | 1600 | 1800 | 2000 | 2200 | 2400 | 2600 | 2700 | 2800 | 3100 | 3300  | 3500  | 3700  | 3800  | 3900  | 4200  |
|                              | $\frac{1}{8}$ | 1800               | 2100 | 2300 | 2400 | 2700 | 3000 | 3300 | 3600 | 3800 | 4000 | 4300 | 4500  | 4800  | 5100  | 5200  | 5400  | 6700  |
|                              | $\frac{1}{4}$ | 2200               | 2600 | 2800 | 3000 | 3400 | 3700 | 4100 | 4400 | 4700 | 4900 | 5200 | 5500  | 6000  | 6400  | 6600  | 6700  | 8400  |
|                              | $\frac{3}{8}$ | 2700               | 3200 | 3400 | 3600 | 4100 | 4500 | 5000 | 5400 | 5700 | 5900 | 6300 | 6800  | 7300  | 7500  | 7900  | 8200  | 10200 |
| 5                            | 0             | 1400               | 1700 | 1800 | 1900 | 2100 | 2400 | 2600 | 2900 | 3000 | 3100 | 3300 | 3500  | 3800  | 4000  | 4100  | 4300  | 5300  |
|                              | $\frac{1}{8}$ | 2000               | 2300 | 2500 | 2600 | 2900 | 3200 | 3600 | 3900 | 4000 | 4200 | 4500 | 4800  | 5200  | 5500  | 5700  | 5800  | 7300  |
|                              | $\frac{1}{4}$ | 2400               | 2800 | 3100 | 3300 | 3700 | 4100 | 4500 | 4800 | 5100 | 5300 | 5700 | 6000  | 6500  | 6900  | 7200  | 7400  | 9200  |
|                              | $\frac{3}{8}$ | 3000               | 3500 | 3700 | 4000 | 4500 | 5000 | 5500 | 6000 | 6200 | 6500 | 6900 | 7400  | 7900  | 8400  | 8700  | 8900  | 11100 |
| 5                            | 6             | 1500               | 1800 | 1900 | 2000 | 2300 | 2500 | 2800 | 3100 | 3200 | 3300 | 3500 | 3800  | 4000  | 4300  | 4400  | 4500  | 5600  |
|                              | $\frac{1}{8}$ | 2100               | 2400 | 2700 | 2800 | 3100 | 3500 | 3800 | 4200 | 4400 | 4600 | 4900 | 5200  | 5600  | 6000  | 6100  | 6200  | 7800  |
|                              | $\frac{1}{4}$ | 2600               | 3100 | 3300 | 3600 | 4000 | 4400 | 4900 | 5300 | 5500 | 5800 | 6200 | 6500  | 7100  | 7500  | 7800  | 8000  | 10000 |
|                              | $\frac{3}{8}$ | 3200               | 3800 | 4000 | 4300 | 4900 | 5400 | 5900 | 6400 | 6700 | 7000 | 7500 | 8100  | 8600  | 9100  | 9400  | 9700  | 12100 |
| 6                            | 0             | 1600               | 1900 | 2000 | 2100 | 2500 | 2700 | 3000 | 3300 | 3400 | 3500 | 3800 | 4100  | 4300  | 4600  | 4700  | 4900  | 6100  |
|                              | $\frac{1}{8}$ | 2300               | 2600 | 2900 | 3000 | 3400 | 3800 | 4100 | 4500 | 4700 | 4900 | 5200 | 5600  | 6000  | 6400  | 6600  | 6800  | 8500  |
|                              | $\frac{1}{4}$ | 2800               | 3300 | 3500 | 3800 | 4300 | 4800 | 5300 | 5700 | 6000 | 6200 | 6700 | 7100  | 7600  | 8100  | 8400  | 8600  | 10800 |
|                              | $\frac{3}{8}$ | 3500               | 4100 | 4400 | 4600 | 5300 | 5800 | 6400 | 7000 | 7300 | 7600 | 8100 | 8700  | 9300  | 9800  | 10200 | 10500 | 13200 |
| 6                            | 6             | 1700               | 2000 | 2200 | 2300 | 2800 | 3000 | 3300 | 3600 | 3700 | 3800 | 4000 | 4300  | 4600  | 4900  | 5000  | 5200  | 6500  |
|                              | $\frac{1}{8}$ | 2500               | 2800 | 3100 | 3200 | 3600 | 4000 | 4400 | 4800 | 5000 | 5200 | 5600 | 6000  | 6400  | 6800  | 7100  | 7300  | 9000  |
|                              | $\frac{1}{4}$ | 3000               | 3500 | 3800 | 4100 | 4600 | 5100 | 5600 | 6100 | 6400 | 6700 | 7200 | 7500  | 8200  | 8700  | 9000  | 9200  | 11500 |
|                              | $\frac{3}{8}$ | 3800               | 4400 | 4700 | 5000 | 5600 | 6300 | 6900 | 7500 | 7800 | 8200 | 8700 | 9400  | 10000 | 10600 | 10900 | 11200 | 14100 |
| 7                            | 0             | 1800               | 2100 | 2300 | 2500 | 2800 | 3000 | 3400 | 3700 | 3800 | 4000 | 4300 | 4600  | 4900  | 5200  | 5300  | 5500  | 6900  |
|                              | $\frac{1}{8}$ | 2600               | 3000 | 3300 | 3400 | 3800 | 4300 | 4700 | 5100 | 5300 | 5600 | 6000 | 6400  | 6800  | 7300  | 7500  | 7700  | 9600  |
|                              | $\frac{1}{4}$ | 3200               | 3800 | 4100 | 4400 | 4900 | 5400 | 6000 | 6500 | 6800 | 7100 | 7600 | 8000  | 8700  | 9300  | 9600  | 10400 | 12800 |
|                              | $\frac{3}{8}$ | 4000               | 4700 | 5000 | 5200 | 6000 | 6700 | 7300 | 8000 | 8400 | 8700 | 9300 | 10000 | 10700 | 11300 | 11700 | 12400 | 15500 |
| 7                            | 6             | 1900               | 2300 | 2400 | 2600 | 2900 | 3200 | 3600 | 3900 | 4100 | 4200 | 4500 | 4800  | 5100  | 5500  | 5600  | 5800  | 7300  |
|                              | $\frac{1}{8}$ | 2800               | 3200 | 3500 | 3600 | 4000 | 4500 | 5000 | 5400 | 5600 | 5800 | 6300 | 6800  | 7300  | 7700  | 8000  | 8200  | 10200 |
|                              | $\frac{1}{4}$ | 3400               | 4000 | 4400 | 4700 | 5200 | 5800 | 6400 | 7000 | 7300 | 7600 | 8100 | 8600  | 9200  | 9800  | 10200 | 10500 | 13100 |
|                              | $\frac{3}{8}$ | 4300               | 5000 | 5300 | 5500 | 6400 | 7100 | 7800 | 8500 | 8900 | 9200 | 9900 | 10700 | 11400 | 12000 | 12400 | 12800 | 16000 |

|   |   |      |      |      |      |      |      |      |      |      |      |      |      |      |      |      |      |      |      |      |      |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |       |    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52400 | 52800 | 53200 | 53600 | 54000 | 54400 | 54800 | 55200 | 55600 | 56000 | 56400 | 56800 | 57200 | 57600 | 58000 | 58400 | 58800 | 59200 | 59600 | 60000 | 60400 | 60800 | 61200 | 61600 | 62000 | 62400 | 62800 | 63200 | 63600 | 64000 | 64400 | 64800 | 65200 | 65600 | 66000 | 66400 | 66800 | 67200 | 67600 | 68000 | 68400 | 68800 | 69200 | 69600 | 70000 | 70400 | 70800 | 71200 | 71600 | 72000 | 72400 | 72800 | 73200 | 73600 | 74000 | 74400 | 74800 | 75200 | 75600 | 76000 | 76400 | 76800 | 77200 | 77600 | 78000 | 78400 | 78800 | 79200 | 79600 | 80000 | 80400 | 80800 | 81200 | 81600 | 82000 | 82400 | 82800 | 83200 | 83600 | 84000 | 84400 | 84800 | 85200 | 85600 | 86000 | 86400 | 86800 | 87200 | 87600 | 88000 | 88400 | 88800 | 89200 | 89600 | 90000 | 90400 | 90800 | 91200 | 91600 | 92000 | 92400 | 92800 | 93200 | 93600 | 94000 | 94400 | 94800 | 95200 | 95600 | 96000 | 96400 | 96800 | 97200 | 97600 | 98000 | 98400 | 98800 | 99200 | 99600 | 100000 | 100400 | 100800 | 101200 | 101600 | 102000 | 102400 | 102800 | 103200 | 103600 | 104000 | 104400 | 104800 | 105200 | 105600 | 106000 | 106400 | 106800 | 107200 | 107600 | 108000 | 108400 | 108800 | 109200 | 109600 | 110000 | 110400 | 110800 | 111200 | 111600 | 112000 | 112400 | 112800 | 113200 | 113600 | 114000 | 114400 | 114800 | 115200 | 115600 | 116000 | 116400 | 116800 | 117200 | 117600 | 118000 | 118400 | 118800 | 119200 | 119600 | 120000 | 120400 | 120800 | 121200 | 121600 | 122000 | 122400 | 122800 | 123200 | 123600 | 124000 | 124400 | 124800 | 125200 | 125600 | 126000 | 126400 | 126800 | 127200 | 127600 | 128000 | 128400 | 128800 | 129200 | 129600 | 130000 | 130400 | 130800 | 131200 | 131600 | 132000 | 132400 | 132800 | 133200 | 133600 | 134000 | 134400 | 134800 | 135200 | 135600 | 136000 | 136400 | 136800 | 137200 | 137600 | 138000 | 138400 | 138800 | 139200 | 139600 | 140000 | 140400 | 140800 | 141200 | 141600 | 142000 | 142400 | 142800 | 143200 | 143600 | 144000 | 144400 | 144800 | 145200 | 145600 | 146000 | 146400 | 146800 | 147200 | 147600 | 148000 | 148400 | 148800 | 149200 | 149600 | 150000 | 150400 | 150800 | 151200 | 151600 | 152000 | 152400 | 152800 | 153200 | 153600 | 154000 | 154400 | 154800 | 155200 | 155600 | 156000 | 156400 | 156800 | 157200 | 157600 | 158000 | 158400 | 158800 | 159200 | 159600 | 160000 | 160400 | 160800 | 161200 | 161600 | 162000 | 162400 | 162800 | 163200 | 163600 | 164000 | 164400 | 164800 | 165200 | 165600 | 166000 | 166400 | 166800 | 167200 | 167600 | 168000 | 168400 | 168800 | 169200 | 169600 | 170000 | 170400 | 170800 | 171200 | 171600 | 172000 | 172400 | 172800 | 173200 | 173600 | 174000 | 174400 | 174800 | 175200 | 175600 | 176000 | 176400 | 176800 | 177200 | 177600 | 178000 | 178400 | 178800 | 179200 | 179600 | 180000 | 180400 | 180800 | 181200 | 181600 | 182000 | 182400 | 182800 | 183200 | 183600 | 184000 | 184400 | 184800 | 185200 | 185600 | 186000 | 186400 | 186800 | 187200 | 187600 | 188000 | 188400 | 188800 | 189200 | 189600 | 190000 | 190400 | 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235200 | 235600 | 236000 | 236400 | 236800 | 237200 | 237600 | 238000 | 238400 | 238800 | 239200 | 239600 | 240000 | 240400 | 240800 | 241200 | 241600 | 242000 | 242400 | 242800 | 243200 | 243600 | 244000 | 244400 | 244800 | 245200 | 245600 | 246000 | 246400 | 246800 | 247200 | 247600 | 248000 | 248400 | 248800 | 249200 | 249600 | 250000 | 250400 | 250800 | 251200 | 251600 | 252000 | 252400 | 252800 | 253200 | 253600 | 254000 | 254400 | 254800 | 255200 | 255600 | 256000 | 256400 | 256800 | 257200 | 257600 | 258000 | 258400 | 258800 | 259200 | 259600 | 260000 | 260400 | 260800 | 261200 | 261600 | 262000 | 262400 | 262800 | 263200 | 263600 | 264000 | 264400 | 264800 | 265200 | 265600 | 266000 | 266400 | 266800 | 267200 | 267600 | 268000 | 268400 | 268800 | 269200 | 269600 | 270000 | 270400 | 270800 | 271200 | 271600 | 272000 | 272400 | 272800 | 273200 | 273600 | 274000 | 274400 | 274800 | 275200 | 275600 | 276000 | 276400 | 276800 | 277200 | 277600 | 278000 | 278400 | 278800 | 279200 | 279600 | 280000 | 280400 | 280800 | 281200 | 281600 | 282000 | 282400 | 282800 | 283200 | 283600 | 284000 | 284400 | 284800 | 285200 | 285600 | 286000 | 286400 | 286800 | 287200 | 287600 | 288000 | 288400 | 288800 | 289200 | 289600 | 290000 | 290400 | 290800 | 291200 | 291600 | 292000 | 292400 | 292800 | 293200 | 293600 | 294000 | 294400 | 294800 | 295200 | 295600 | 296000 | 296400 | 296800 | 297200 | 297600 | 298000 | 298400 | 298800 | 299200 | 299600 | 300000 | 300400 | 300800 | 301200 | 301600 | 302000 | 302400 | 302800 | 303200 | 303600 | 304000 | 304400 | 304800 | 305200 | 305600 | 306000 | 306400 | 306800 | 307200 | 307600 | 308000 | 308400 | 308800 | 309200 | 309600 | 310000 | 310400 | 310800 | 311200 | 311600 | 312000 | 312400 | 312800 | 313200 | 313600 | 314000 | 314400 | 314800 | 315200 | 315600 | 316000 | 316400 | 316800 | 317200 | 317600 | 318000 | 318400 | 318800 | 319200 | 319600 | 320000 | 320400 | 320800 | 321200 | 321600 | 322000 | 322400 | 322800 | 323200 | 323600 | 324000 | 324400 | 324800 | 325200 | 325600 | 326000 | 326400 | 326800 | 327200 | 327600 | 328000 | 328400 | 328800 | 329200 | 329600 | 330000 | 330400 | 330800 | 331200 | 331600 | 332000 | 332400 | 332800 | 333200 | 333600 | 334000 | 334400 | 334800 | 335200 | 335600 | 336000 | 336400 | 336800 | 337200 | 337600 | 338000 | 338400 | 338800 | 339200 | 339600 | 340000 | 340400 | 340800 | 341200 | 341600 | 342000 | 342400 | 342800 | 343200 | 343600 | 344000 | 344400 | 344800 | 345200 | 345600 | 346000 | 346400 | 346800 | 347200 | 347600 | 348000 | 348400 | 348800 | 349200 | 349600 | 350000 | 350400 | 350800 | 351200 | 351600 | 352000 | 352400 | 352800 | 353200 | 353600 | 354000 | 354400 | 354800 | 355200 | 355600 | 356000 | 356400 | 356800 | 357200 | 357600 | 358000 | 358400 | 358800 | 359200 | 359600 | 360000 | 360400 | 360800 | 361200 | 361600 | 362000 | 362400 | 362800 | 363200 | 363600 | 364000 | 364400 | 364800 | 365200 | 365600 | 366000 | 366400 | 366800 | 367200 | 367600 | 368000 | 368400 | 368800 | 369200 | 369600 | 370000 | 370400 | 370800 | 371200 | 371600 | 372000 | 372400 | 372800 | 373200 | 373600 | 374000 | 374400 | 374800 | 375200 | 375600 | 376000 | 376400 | 376800 | 377200 | 377600 | 378000 | 378400 | 378800 | 379200 | 379600 | 380000 | 380400 | 380800 | 381200 | 381600 | 382000 | 382400 | 382800 | 383200 | 383600 | 384000 | 384400 | 384800 | 385200 | 385600 | 386000 | 386400 | 386800 | 387200 | 387600 | 388000 | 388400 | 388800 | 389200 | 389600 | 390000 | 390400 | 390800 | 391200 | 391600 | 392000 | 392400 | 392800 | 393200 | 393600 | 394000 | 394400 | 394800 | 395200 | 395600 | 396000 | 396400 | 396800 | 397200 | 397600 | 398000 | 398400 | 398800 | 399200 | 399600 | 400000 | 400400 | 400800 | 401200 | 401600 | 402000 | 402400 | 402800 | 403200 | 403600 | 404000 | 404400 | 404800 | 405200 | 405600 | 406000 | 406400 | 406800 | 407200 | 407600 | 408000 | 408400 | 408800 | 409200 | 409600 | 410000 | 410400 | 410800 | 411200 | 411600 | 412000 | 412400 | 412800 | 413200 | 413600 | 414000 | 414400 | 414800 | 415200 | 415600 | 416000 | 416400 | 416800 | 417200 | 417600 | 418000 | 418400 | 418800 | 419200 | 419600 | 420000 | 420400 | 420800 | 421200 | 421600 | 422000 | 422400 | 422800 | 423200 | 423600 | 424000 | 424400 | 424800 | 425200 | 425600 | 426000 | 426400 | 426800 | 427200 | 427600 | 428000 | 428400 | 428800 | 429200 | 429600 | 430000 | 430400 | 430800 | 431200 | 431600 | 432000 | 432400 | 432800 | 433200 | 433600 | 434000 | 434400 | 434800 | 435200 | 435600 | 436000 | 436400 | 436800 | 437200 | 437600 | 438000 | 438400 | 438800 | 439200 | 439600 | 440000 | 440400 | 440800 | 441200 | 441600 | 442000 | 442400 | 442800 | 443200 | 443600 | 444000 | 444400 | 444800 | 445200 | 445600 | 446000 | 446400 | 446800 | 447200 | 447600 | 448000 | 448400 | 448800 | 449200 | 449600 | 450000 | 450400 | 450800 | 451200 | 451600 | 452000 | 452400 | 452800 | 453200 | 453600 | 454000 | 454400 | 454800 | 455200 | 455600 | 456000 | 456400 | 456 |
|---|---|------|------|------|------|------|------|------|------|------|------|------|------|------|------|------|------|------|------|------|------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|--------|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TABLE XLVIII.—continued.

| DIAMETERS IN FEET. |                      |               |               |               |               |               |               |               |       |                |                |                |                |                |                |                |       |                |                |                |       |
|--------------------|----------------------|---------------|---------------|---------------|---------------|---------------|---------------|---------------|-------|----------------|----------------|----------------|----------------|----------------|----------------|----------------|-------|----------------|----------------|----------------|-------|
|                    |                      | 12            | 14            | 15            | 16            | 18            | 20            | 22            | 24    | 25             | 26             | 28             | 30             | 32             | 34             | 35             | 36    | 38             | 40             | 45             |       |
| Depth of Sides.    | Thickness of Plates. |               |               |               |               |               |               |               |       |                |                |                |                |                |                |                |       |                |                |                |       |
|                    |                      | $\frac{1}{8}$ | $\frac{1}{4}$ | $\frac{3}{8}$ | $\frac{1}{2}$ | $\frac{5}{8}$ | $\frac{3}{4}$ | $\frac{7}{8}$ | 1     | $1\frac{1}{8}$ | $1\frac{1}{4}$ | $1\frac{3}{8}$ | $1\frac{1}{2}$ | $1\frac{5}{8}$ | $1\frac{3}{4}$ | $1\frac{7}{8}$ | 2     | $2\frac{1}{8}$ | $2\frac{1}{4}$ | $2\frac{3}{8}$ |       |
| 12                 | 6                    | 2900          | 3500          | 3700          | 4000          | 4500          | 4900          | 5400          | 6000  | 6800           | 6400           | 6900           | 7400           | 7800           | 8400           | 8600           | 8900  | 9400           | 9800           | 11800          |       |
|                    |                      | 4400          | 5000          | 5400          | 5600          | 6300          | 7100          | 7800          | 8500  | 9800           | 9800           | 9900           | 10700          | 11300          | 12100          | 12900          | 13500 | 14300          | 14800          | 15000          |       |
|                    |                      | 5400          | 6400          | 7000          | 7500          | 8300          | 9200          | 10200         | 11000 | 11600          | 12100          | 12900          | 13500          | 14800          | 15600          | 16200          | 16700 | 17500          | 18500          | 20900          |       |
|                    |                      | 6800          | 8000          | 8500          | 9100          | 10300         | 11400         | 12500         | 13800 | 14300          | 14900          | 15900          | 17200          | 18300          | 19200          | 19900          | 20500 | 21700          | 22900          | 25700          |       |
| 13                 | 0                    | 3000          | 3600          | 3800          | 4100          | 4700          | 5100          | 5600          | 6200  | 6900           | 6500           | 6800           | 7100           | 7700           | 8100           | 8700           | 8900  | 9200           | 10200          | 11700          |       |
|                    |                      | 4500          | 5100          | 5500          | 5800          | 6600          | 7400          | 8000          | 8800  | 9200           | 9700           | 10300          | 11100          | 11700          | 12600          | 13300          | 13900 | 14500          | 15300          | 16600          |       |
|                    |                      | 5600          | 6700          | 7200          | 7800          | 8600          | 9500          | 10600         | 11400 | 12000          | 12500          | 13400          | 14000          | 15300          | 16200          | 16900          | 17500 | 18200          | 19200          | 21700          |       |
|                    |                      | 7200          | 8300          | 8800          | 9400          | 10300         | 11300         | 12400         | 13500 | 14300          | 14800          | 15400          | 16500          | 17800          | 19000          | 19900          | 20700 | 21400          | 22600          | 26700          |       |
| 13                 | 6                    | 3100          | 3700          | 4000          | 4300          | 4900          | 5300          | 5800          | 6400  | 6700           | 6900           | 7000           | 8000           | 8400           | 8900           | 9200           | 9500  | 10100          | 10500          | 12100          |       |
|                    |                      | 4600          | 5200          | 5700          | 6000          | 6800          | 7600          | 8200          | 9000  | 9500           | 10000          | 10600          | 11500          | 12100          | 13000          | 13500          | 13800 | 14500          | 15300          | 17200          |       |
|                    |                      | 5700          | 6800          | 7300          | 7900          | 8700          | 9600          | 10700         | 11800 | 12400          | 13000          | 13800          | 14500          | 15800          | 16800          | 17400          | 17900 | 18800          | 19900          | 22500          |       |
|                    |                      | 7400          | 8500          | 9000          | 9700          | 11100         | 12300         | 13400         | 14500 | 15400          | 16000          | 16600          | 17100          | 18500          | 19700          | 20600          | 21400 | 22100          | 23400          | 24500          | 27700 |
| 14                 | 0                    | 3200          | 3800          | 4100          | 4400          | 5000          | 5400          | 6000          | 6600  | 6900           | 7100           | 7000           | 8200           | 8600           | 9100           | 9500           | 9800  | 10400          | 10800          | 12500          |       |
|                    |                      | 4700          | 5300          | 5800          | 6100          | 6900          | 7700          | 8300          | 9100  | 9600           | 10300          | 11000          | 11900          | 12500          | 13400          | 14000          | 14300 | 15000          | 15800          | 17700          |       |
|                    |                      | 5800          | 6900          | 7400          | 8000          | 8800          | 9700          | 10800         | 11900 | 12500          | 13000          | 13800          | 14400          | 15700          | 16700          | 17300          | 17800 | 18500          | 19500          | 22300          |       |
|                    |                      | 7700          | 8800          | 9300          | 10100         | 11500         | 12700         | 13800         | 15000 | 15900          | 16000          | 17000          | 18400          | 19600          | 20400          | 21400          | 22200 | 23500          | 24600          | 28000          |       |
| 14                 | 6                    | 3300          | 3900          | 4200          | 4500          | 5100          | 5500          | 6100          | 6700  | 7100           | 7300           | 7900           | 8500           | 8900           | 9400           | 9800           | 10100 | 10800          | 11200          | 12900          |       |
|                    |                      | 4800          | 5400          | 5900          | 6200          | 7000          | 7800          | 8400          | 9200  | 9700           | 10400          | 11100          | 12000          | 12600          | 13500          | 14100          | 14400 | 15100          | 16000          | 18000          |       |
|                    |                      | 5900          | 7000          | 7500          | 8100          | 8900          | 9800          | 10900         | 12000 | 12600          | 13300          | 13900          | 14800          | 15500          | 16400          | 17000          | 17500 | 18200          | 19100          | 21300          |       |
|                    |                      | 7900          | 9000          | 9500          | 10300         | 11700         | 12900         | 14000         | 15100 | 16000          | 16100          | 17100          | 18500          | 19700          | 20500          | 21500          | 22300 | 23600          | 24700          | 28100          |       |
| 15                 | 0                    | 3400          | 4100          | 4400          | 4700          | 5300          | 5700          | 6300          | 6900  | 7400           | 7600           | 8100           | 8700           | 9200           | 9600           | 10000          | 10300 | 11000          | 11500          | 13300          |       |
|                    |                      | 4900          | 5500          | 6000          | 6300          | 7100          | 7900          | 8500          | 9300  | 9800           | 10500          | 11200          | 12100          | 12700          | 13600          | 14200          | 14500 | 15200          | 16100          | 18100          |       |
|                    |                      | 6000          | 7100          | 7600          | 8200          | 9000          | 9900          | 11000         | 12100 | 12700          | 13400          | 14000          | 15000          | 15600          | 16500          | 17100          | 17600 | 18300          | 19200          | 21500          |       |
|                    |                      | 8000          | 9100          | 9600          | 10400         | 11800         | 13000         | 14100         | 15200 | 16100          | 16200          | 17200          | 18600          | 19800          | 20600          | 21600          | 22400 | 23700          | 24800          | 28200          |       |
| 15                 | 6                    | 3500          | 4200          | 4500          | 4800          | 5400          | 5800          | 6400          | 7000  | 7500           | 7700           | 8200           | 8800           | 9300           | 9700           | 10100          | 10400 | 11100          | 11600          | 13400          |       |
|                    |                      | 5000          | 5600          | 6100          | 6400          | 7200          | 8000          | 8600          | 9400  | 9900           | 10600          | 11300          | 12200          | 12800          | 13700          | 14300          | 14600 | 15300          | 16200          | 18200          |       |
|                    |                      | 6100          | 7200          | 7700          | 8300          | 9100          | 10000         | 11100         | 12200 | 12800          | 13500          | 14100          | 15100          | 15700          | 16600          | 17200          | 17700 | 18400          | 19300          | 21600          |       |
|                    |                      | 8100          | 9200          | 9700          | 10500         | 11900         | 13100         | 14200         | 15300 | 16200          | 16300          | 17300          | 18700          | 19900          | 20700          | 21700          | 22500 | 23800          | 24900          | 28300          |       |
| 16                 | 0                    | 3600          | 4300          | 4600          | 4900          | 5500          | 5900          | 6500          | 7100  | 7600           | 7800           | 8300           | 8900           | 9400           | 9800           | 10200          | 10500 | 11200          | 11700          | 13500          |       |
|                    |                      | 5100          | 5700          | 6200          | 6500          | 7300          | 8100          | 8700          | 9500  | 10000          | 10700          | 11400          | 12300          | 12900          | 13800          | 14400          | 14700 | 15400          | 16300          | 18300          |       |
|                    |                      | 6200          | 7300          | 7800          | 8400          | 9200          | 10100         | 11200         | 12300 | 12900          | 13600          | 14200          | 15200          | 15800          | 16700          | 17300          | 17800 | 18500          | 19400          | 21700          |       |
|                    |                      | 8200          | 9300          | 9800          | 10600         | 12000         | 13200         | 14300         | 15400 | 16300          | 16400          | 17400          | 18800          | 20000          | 20800          | 21800          | 22600 | 23900          | 25000          | 28400          |       |

TABLE XLIX.—Approximate Weight in Pounds of Material in Bottoms of Steel Vats, including Rivets and Allowance for Laps.

| Thickness of Plates | DIAMETERS IN FEET. |      |      |      |      |      |      |      |      |      |       |       |       |       |       |       |
|---------------------|--------------------|------|------|------|------|------|------|------|------|------|-------|-------|-------|-------|-------|-------|
|                     | 12                 | 14   | 15   | 16   | 18   | 20   | 22   | 24   | 25   | 26   | 28    | 30    | 32    | 34    | 35    | 36    |
| $\frac{1}{8}$       | 600                | 900  | 1000 | 1100 | 1400 | 1700 | 2100 | 2500 | 2800 | 3000 | 3400  | 3800  | 4400  | 5000  | 5300  | 5600  |
| $\frac{1}{4}$       | 900                | 1400 | 1500 | 1700 | 2100 | 2600 | 3300 | 3800 | 4200 | 4400 | 5100  | 5900  | 6600  | 7500  | 7900  | 8400  |
| $\frac{3}{8}$       | 1300               | 1800 | 2000 | 2300 | 2800 | 3400 | 4200 | 5000 | 5600 | 5800 | 6800  | 7800  | 8800  | 10000 | 10600 | 11000 |
| $\frac{1}{2}$       | 1600               | 2300 | 2500 | 2900 | 3500 | 4300 | 5200 | 6300 | 7000 | 7300 | 8500  | 9600  | 11000 | 12300 | 13200 | 14000 |
| $\frac{3}{4}$       | 1800               | 2700 | 3000 | 3300 | 4200 | 5100 | 6200 | 7500 | 8400 | 8700 | 10200 | 11700 | 13200 | 15000 | 15900 | 16800 |

TABLE L.—Approximate Number of Rivets in Vats.

| Depth of Vats.<br>Ft. Ins. | DIAMETERS IN FEET. |      |      |      |      |      |      |      |      |      |      |      |      |      |      |      |
|----------------------------|--------------------|------|------|------|------|------|------|------|------|------|------|------|------|------|------|------|
|                            | 12                 | 14   | 15   | 16   | 18   | 20   | 22   | 24   | 25   | 26   | 28   | 30   | 32   | 34   | 35   | 36   |
| 4 0                        | 1000               | 1200 | 1400 | 1500 | 1800 | 2000 | 2200 | 2500 | 2600 | 2800 | 3100 | 3500 | 3900 | 4300 | 4400 | 4600 |
| 4 6                        | 1000               | 1200 | 1400 | 1500 | 1800 | 2100 | 2300 | 2500 | 2600 | 2900 | 3200 | 3600 | 3900 | 4300 | 4500 | 4700 |
| 5 0                        | 1100               | 1300 | 1500 | 1600 | 1900 | 2100 | 2300 | 2600 | 2700 | 2900 | 3200 | 3600 | 3900 | 4400 | 4600 | 4800 |
| 5 6                        | 1100               | 1300 | 1500 | 1600 | 1900 | 2100 | 2300 | 2600 | 2700 | 2900 | 3200 | 3600 | 4000 | 4400 | 4600 | 4800 |
| 6 0                        | 1200               | 1400 | 1600 | 1700 | 2000 | 2200 | 2400 | 2700 | 2800 | 3000 | 3300 | 3700 | 4000 | 4500 | 4700 | 4900 |
| 6 6                        | 1200               | 1400 | 1600 | 1700 | 2000 | 2200 | 2400 | 2700 | 2800 | 3000 | 3300 | 3700 | 4000 | 4500 | 4700 | 4900 |
| 7 0                        | 1300               | 1500 | 1700 | 1800 | 2100 | 2300 | 2500 | 2800 | 2900 | 3100 | 3400 | 3800 | 4100 | 4600 | 4800 | 5000 |
| 7 6                        | 1300               | 1500 | 1700 | 1800 | 2100 | 2300 | 2500 | 2800 | 2900 | 3100 | 3400 | 3800 | 4100 | 4600 | 4800 | 5000 |
| 8 0                        | 1300               | 1500 | 1700 | 1800 | 2100 | 2300 | 2500 | 2800 | 2900 | 3100 | 3400 | 3800 | 4100 | 4600 | 4800 | 5000 |
| 8 6                        | 1300               | 1500 | 1700 | 1800 | 2100 | 2300 | 2500 | 2800 | 2900 | 3100 | 3400 | 3800 | 4100 | 4600 | 4800 | 5000 |
| 9 0                        | 1300               | 1500 | 1700 | 1800 | 2100 | 2300 | 2500 | 2800 | 2900 | 3100 | 3400 | 3800 | 4100 | 4600 | 4800 | 5000 |
| 9 6                        | 1400               | 1600 | 1800 | 1900 | 2200 | 2400 | 2600 | 2900 | 3000 | 3200 | 3500 | 3900 | 4200 | 4700 | 4900 | 5100 |
| 10 0                       | 1400               | 1600 | 1800 | 1900 | 2200 | 2400 | 2600 | 2900 | 3000 | 3200 | 3500 | 3900 | 4200 | 4700 | 4900 | 5100 |
| 10 6                       | 1400               | 1600 | 1800 | 1900 | 2200 | 2400 | 2600 | 2900 | 3000 | 3200 | 3500 | 3900 | 4200 | 4700 | 4900 | 5100 |
| 11 0                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 11 6                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 12 0                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 12 6                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 13 0                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 13 6                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 14 0                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 14 6                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 15 0                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 15 6                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |
| 16 0                       | 1500               | 1700 | 1900 | 2000 | 2300 | 2500 | 2700 | 3000 | 3100 | 3300 | 3600 | 4000 | 4300 | 4800 | 5000 | 5200 |

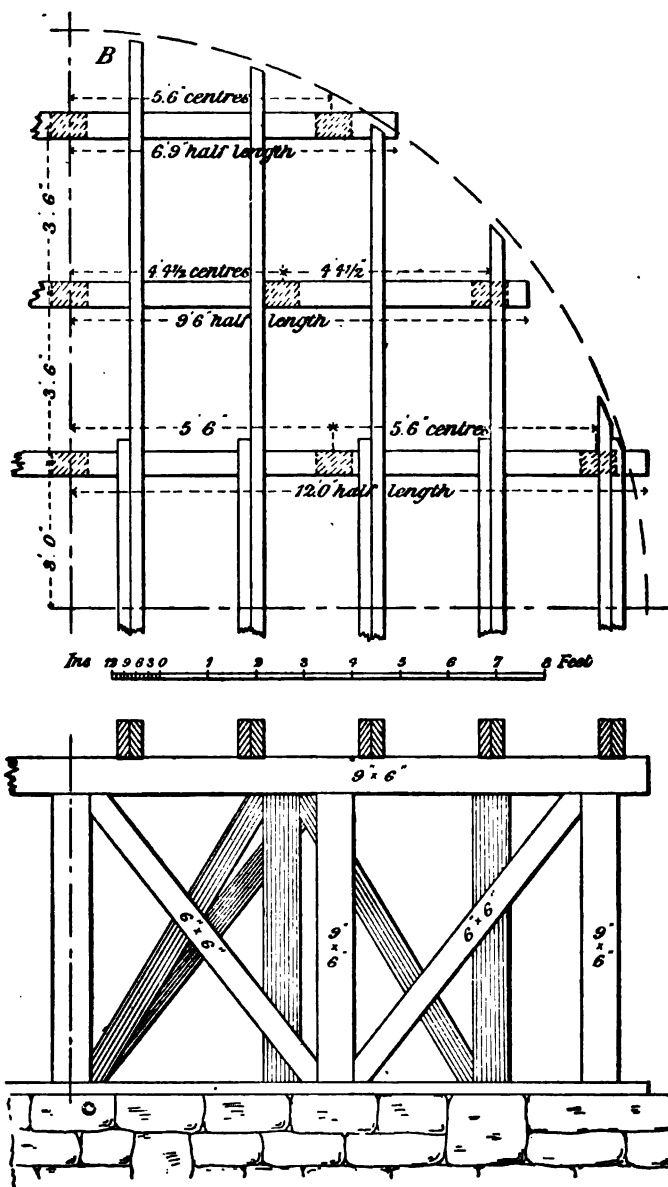


FIG. 66.—Timber Foundation Frame (Plan and Section).

The following table gives average quantities of timber in the two kinds of supports above described :

TABLE LI.—Quantities in Timber Supports, exclusive of Joists.

| Diameter of Vats in feet. | As in Fig. 65.              |           | As in Fig. 66               |           |
|---------------------------|-----------------------------|-----------|-----------------------------|-----------|
|                           | Square feet one inch thick. | Cu. feet. | Square feet one inch thick. | Cu. feet. |
| 16                        | 684                         | 57        | 1389                        | 116       |
| 18                        | 850                         | 71        | 1488                        | 124       |
| 20                        | 920                         | 77        | 1646                        | 137       |
| 22                        | 1100                        | 92        | 1836                        | 153       |
| 24                        | 1183                        | 99        | 2161                        | 180       |

**Masonry Foundations for Vats.**—These are almost invariably built in hammer-dressed rubble masonry, worked to level courses about every 2 ft.

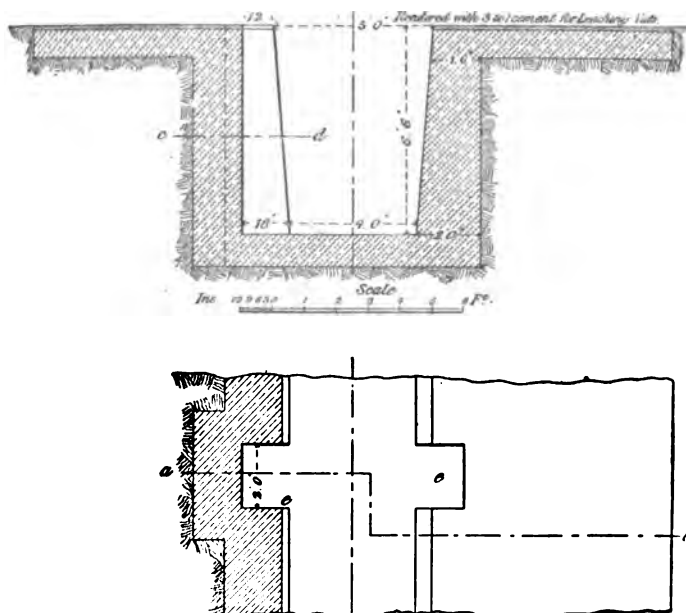


FIG. 67.—Masonry-lined Tunnel ; Section on *ab* and *cd* and Half Plan.

of height, set in good hydraulic lime, and pointed with Portland cement on exposed faces. When the bottom of the vat is about level with the surface of the ground, so that the discharge tunnel has to be excavated, the type shown in Fig. 67 will be found suitable. The minimum clear height in the tunnel

should be 6 ft. to allow men to work comfortably, and therefore efficiently. A fall of at least 1 in 100 should be given to the tunnel floor for drainage; and, where trucks are used for discharging, refuge recesses, as at *e*, Fig. 67, should be formed at intervals in the side walls. Where rope haulage is used, these recesses should be particularly large and frequent, but it is still better

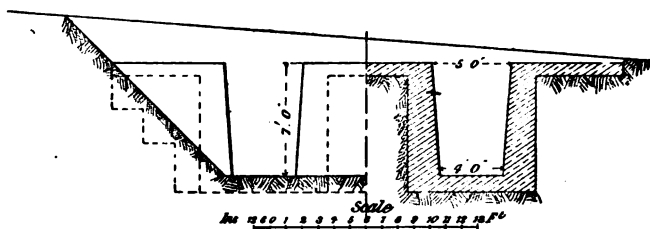


FIG. 68.—Half Elevation at End of Tunnel, and Half Cross-section through Tunnel.

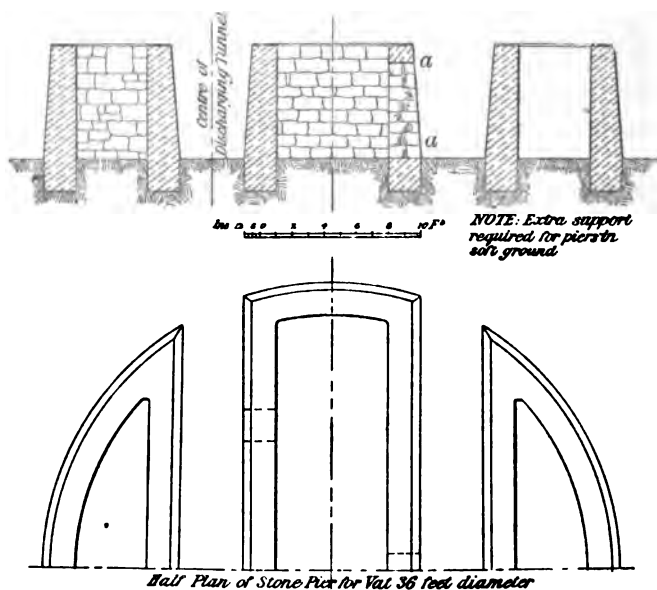


FIG. 69.—Pier for Single-tier Vats; Section and Half Plan.

in this case to make the tunnel of greater width than shown in the drawings, which represent the minimum required with 20-cu. ft. side-tipping trucks, worked by hand. Fig. 68 shows a suitable section for the foundation of a 32-ft. vat, with two tunnels underneath.

When vats have to be supported above the surface of the ground, the neatest arrangement consists of a circular pier, with several inside parallel walls to carry the vat joists, as shown in Fig. 69. The stone work is only carried down 2 ft. below the surface level in Fig. 69, and this depth is also assumed in the quantities given in Table LII. But, of course, in each case

the actual depth is determined by the nature of the ground, as proved by trial excavations, and the importance of this point cannot be overestimated.

TABLE LII.—Quantities in Masonry Foundation in cubic yards for one Vat, when built on Cemented Gravel or other hard soil, two feet below Floor Level.

| Diam. of Vat in feet. | As in Fig. 67. Single Tunnel. | As in Fig. 68. Two Tunnels. | Circular Pier. Single Tunnel. | Circular Pier, as in Fig. 69. Two Tunnels. | Parallel Walls, as in Figs. 70, 71, 72. Single Tunnels. | Parallel Walls, as in Figs. 71, 72. Two Tunnels. |
|-----------------------|-------------------------------|-----------------------------|-------------------------------|--|---|--|
| 20                    | 36.41                         | ...                         | 46.10                         | ...  | ...   | ...  |
| 24                    | 46.75                         | 69.50                       | 60.64                         | ...  | 108.76  | ...  |
| 28                    | 59.21                         | 85.75                       | 88.15                         | 88.15                                      | 181.32  | 165.28   |
| 32                    | 71.99                         | 101.90                      | ...                           | 101.82                                     | ...   | 220.87   |
| 36                    | ...                           | 120.74                      | ...                           | 116.74                                     | ...   | 288.00   |
| 40                    | ...                           | 142.50                      | ...                           | 161.48                                     | ...   | 260.37   |

Circular piers are sometimes built with projecting portions, as in Fig. 70, for carrying the trestles of overhead staging or the iron columns for an upper tier of vats. Another arrangement often adopted consists of a number of straight parallel walls, as in Fig. 71, which is adapted to carry a lower vat

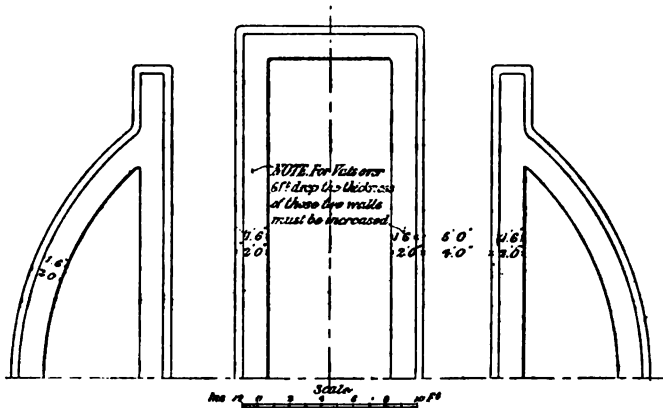


FIG. 70. — Pier with Projections for carrying Staging; Half Plan.

25 ft. in diameter and eight cast-iron columns supporting an upper vat. In this case, properly dressed bedstones *c c* must be built into the walls to receive the bases of the columns.

. When necessary, manholes should be provided to allow access to the under side of the tanks, as in Fig. 69, where two openings for this purpose are shown in plan and one in section *a a*.



Table LII. gives average quantities of masonry for several sizes of vats for some of the arrangements described.

The blank spaces are left because they correspond to conditions which are unlikely to occur in practice. Some diameters lend themselves to the spacing of the walls better than others, hence the apparent discrepancies in the last four columns. The figures are, however, correct, as they have been taken out from drawings made to a scale of  $\frac{1}{4}$  in. to 1 ft.

**Foundations for Pachuca Vats.**—These require special attention on account of the heavy loads concentrated on small areas. For instance, a vat 45 ft. high and 15 ft. diameter, filled with slime at  $1\frac{1}{2}$  to 1 dilution, contains about 300 long tons of pulp, and its own weight is about 14 tons. This load has to be evenly distributed over the foundation.

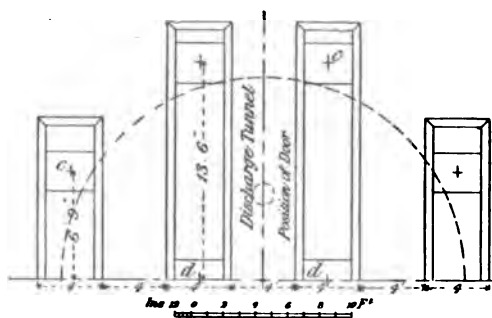


FIG. 71.—Pier for Double-tier Vats; Half Plan.

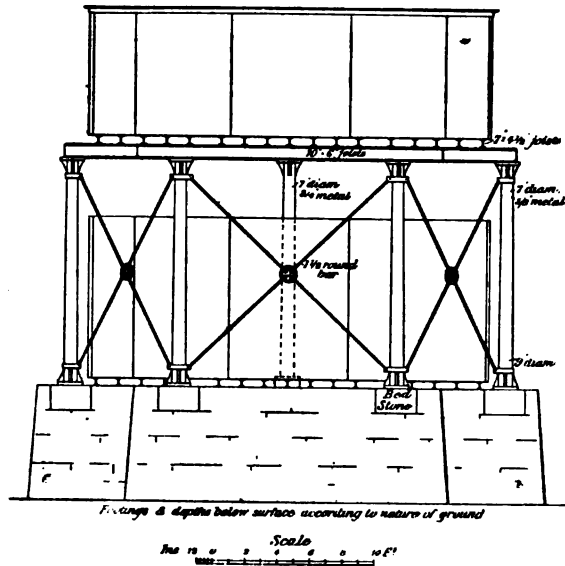
On good ground a thick mass of concrete is sufficient, but where the ground will not carry more than 1 ton per square foot a reinforced concrete mat is to be recommended, in which two sets of parallel steel beams are used. The upper set is laid at right angles to the lower set, and all are completely embedded in the concrete.

These beams may be rolled sections or old rails. Their size may be calculated as in the case of foundation grillages for the columns of buildings, *i.e.* on the assumption that they receive a uniform upward pressure from the ground, and that they are supported at the points which lie under the bottom rings of the vats.

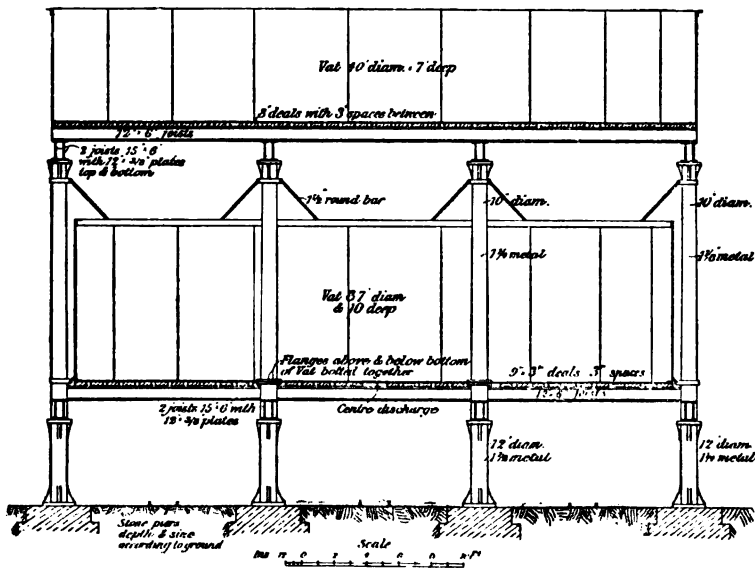
With a single vat, or a number of vats in a single line, a horizontal wind pressure of 25 lb. per square foot gives an additional pressure, on the opposite side of the foundation, equal to 1 ton per square foot. The best way of providing against this is to place the vats in two rows and to connect them together by cross bracing.

**Iron and Steel Supports for Vats.**—When steel vats are used, rolled-steel joists are now always employed to support them, unless old rails or some other cheap substitute happens to be available. In single-tier plants, and also in those with two tiers when the lower vats are carried on masonry walls, only a single set of fairly light joists is required for each vat, as in Fig. 72. But in

other cases these light joists rest upon main girders, which in turn are carried upon columns, as shown in the upper part of Fig. 72 and in Fig. 73.



**FIG. 72.—Pier for Double-tier Vats ; Elevation.**



**FIG. 73.—Pier of Iron Supports for Double-tier Vats ; Sectional Elevation.**

The sizes of the various steel sections available are definitely fixed. The chief point in designing such structures is, therefore, to arrange the spacing

of joists and girders so as to utilise fully the strength of each selected section. As considerable latitude is allowed in the spacing of the girders it is generally possible, by trial of several arrangements, to utilise those sizes which give the greatest strength for the least weight.

It is necessary, first of all, to ascertain the maximum distance allowable between the joists which immediately support the bottom plates of the vat, this distance, of course, depending upon the thickness of the plates, and the load resting upon them. An appropriate result may be obtained by considering only a strip of plate, say 12 in. wide, and calculating as for a solid beam of this width whose depth is equal to the thickness of the plate.

If  $L$  equals the distance in inches between centres of joists,  $d$  equals the depth of vat in feet, and  $t$  equals the thickness of bottom plates in inches, then, from the usual formulæ for beams fixed at ends, and with a limiting stress of 18,000 lb. per square inch, the following formula is obtained :

$$L = \frac{208t}{\sqrt{d}}.$$

From this the maximum distances allowable between centres of joists have been calculated. These are given in Table LIII.

TABLE LIII.—*Maximum Distance in inches between Centres of Joists, according to Thickness of Bottom Plates.*

| Thickness<br>of<br>Plates.<br>Inches. | Depth of Vats in feet. |                 |                 |                 |                 |                 |
|---------------------------------------|------------------------|-----------------|-----------------|-----------------|-----------------|-----------------|
|                                       | 6                      | 8               | 10              | 12              | 14              | 16              |
| $\frac{3}{8}$                         | 16                     | 14              | $12\frac{1}{2}$ | $11\frac{1}{2}$ | $10\frac{1}{2}$ | $9\frac{1}{2}$  |
| $\frac{1}{2}$                         | $21\frac{1}{2}$        | $18\frac{1}{2}$ | $16\frac{1}{2}$ | 15              | 14              | 13              |
| $\frac{5}{8}$                         | $26\frac{1}{2}$        | 23              | $20\frac{1}{2}$ | $18\frac{1}{2}$ | $17\frac{1}{2}$ | $16\frac{1}{2}$ |
| $\frac{3}{4}$                         | $31\frac{1}{2}$        | $27\frac{1}{2}$ | $24\frac{1}{2}$ | $22\frac{1}{2}$ | $20\frac{1}{2}$ | $19\frac{1}{2}$ |

Vats have been erected in which the above figures are exceeded by as much as 25 per cent., and the plates have stood the strain, but in these cases the factor of safety must be less than is desirable for permanent structures. In some cases a floor of 3-in. deals laid on the flat about 3 in. apart is interposed between the vat and the joists. When this is done the latter may be placed at 4-ft. centres for a vat 10 ft. deep. The question whether it pays to use the timber in this way may be settled, under any given conditions as to relative cost of wood and iron, by finding the weight of joists required in either case, in the following manner :

For  $\frac{3}{16}$ -in. plates without timber and with joists at 15-in. centres, the load per joist on an 8-ft. space is 5·4 tons, and a 6-in.  $\times$  2½-in. joist weighing

14 lb. per running foot is sufficient. So in this case the weight of steel per square foot of vat supported thereby is 11.2 pounds.

Similarly :

For  $\frac{1}{4}$ -in. plates, joists weigh 10.2 lb. per square foot of vat.

„  $\frac{1}{8}$ -in. „ „ 10 „ „

With the deal floor and joists at 4-ft. centres, the weight of joists is 8 lb. per square foot of vat. So the saving at the best is only 3.2 lb. of steel against the price of 1 running foot of  $9 \times 3$ -in. deal. At the usual relative prices of the two materials, it is clear that there is little difference in cost either way. Probably with  $\frac{3}{16}$ -in. plates the deals cause a slight economy, but with thicker plates it is cheaper to omit them. Their only advantage is that they afford a more yielding bed for the vat to rest upon.

In laying out the main girders, it is necessary to keep in mind the well-known fact that the total weight of metal in the girders for any given load increases with the span, so that if the columns are too far apart the weight both of main girders and of joists will be unduly great. Table LIV., when the cost per ton of the material is known, enables the designer to determine the extra cost of any proposed increase of span, or *vice versa*. The light sections are applicable to the joists and the heavy sections to the main girders.

TABLE LIV.—*Showing Average Relation between Weight of Rolled-Steel I-beams and the Safe Load carried by them. Stress allowed  $7\frac{1}{2}$  tons per square inch. First column refers to long ton of 2240 lb. The second is applicable to any unit.*

| Span<br>in feet. | Light sections from $4 \times 3$ in.<br>to $10 \times 6$ in. |   | Heavy sections from $10 \times 8$ in.<br>to $24 \times 7\frac{1}{2}$ in. |  |
|------------------|--|---|--|--|
|                  | Weight of Joists<br>in lb. per ton<br>of load.               | Weight of Joists<br>expressed as per-<br>centage of load. | Weight of Girders<br>in lb. per ton<br>of load.                          | Weight of Girders<br>expressed as per-<br>centage of load. |
| 6                | 13   | 0.58  | ..   | ...  |
| 8                | 18.90  | 0.84  | ..   | ...  |
| 10               | 24.40  | 1.09  | ..   | ...  |
| 12               | 32.96  | 1.47  | 22.64  | 1.01   |
| 14               | 44.92  | 2.01  | 29.47  | 1.32   |
| 16               | 55.03  | 2.46  | 36.26  | 1.62   |
| 18               | 64.48  | 2.88  | 42.80  | 1.91   |
| 20               | ...  | ...   | 50.73  | 2.26   |
| 22               | ...  | ...   | 61.71  | 2.75   |
| 24               | ...  | ...   | 68.39  | 3.05   |
| 26               | ...  | ...   | 80.58  | 3.60   |

**Columns.**—Cast iron is generally used, but in cases where transport to the mine is exceptionally difficult, stanchions of rolled-steel sections may be recommended as being less liable to accidental breakage. As shown in Table LV., there is very little difference in the weight of material required in

either case, but cast iron is cheaper per ton. From this table the quantities required for any given load can be readily obtained for estimating purposes.

TABLE LV.—*Relation between Weight of Columns and Loads sustained by them.*

| Height of Column in feet. | Cast-iron Columns.<br>Factor of Safety = 5. |   | Steel Columns.<br>Factor of Safety = 4.  |   |
|---------------------------|---|---|--|---|
|                           | Weight of Column in lb. per ton of load.    | Weight of Column as percentage of load. | Weight of Column in lb. per ton of load. | Weight of Column as percentage of load. |
| 6                         | 4.56  | 0.20                                    | 3.50                                     | 0.16                                    |
| 8                         | 6.02  | 0.27                                    | 5.18                                     | 0.23                                    |
| 10                        | 7.96  | 0.36                                    | 7.36                                     | 0.33                                    |
| 12                        | 10.22                                       | 0.46                                    | 9.96                                     | 0.45                                    |
| 14                        | 13.14                                       | 0.59                                    | 13.25                                    | 0.59                                    |
| 16                        | 16.67                                       | 0.74                                    | 17.82                                    | 0.79                                    |

Fig. 73 shows a pair of superimposed vats, both of which are carried upon cast-iron columns.

In Fig. 72 the masonry walls are the same as those shown in plan in Fig. 71, and the upper and lower vats are each 25 ft. in diameter. It is, however, also quite common, and indeed somewhat better practice when all the vats are in one line, to have the top vat of larger diameter and less depth than the lower one, so that part of its weight comes directly over the columns, as shown in Fig. 73, thus leaving less weight to be supported by the girders. The columns should be stayed by diagonal ties of  $1\frac{1}{2}$  to  $1\frac{3}{4}$ -in. round iron between them, and means must be provided for tightening these when in place, such as by means of the central ring with nuts inside it, shown in the drawings. Holding-down bolts for the columns must be built in the masonry, and the main girders should be connected by plates and bolts where they meet on top of the columns, and the joists also bolted to the girders where they cross the latter over or close to a column, thus binding the whole of the separate parts into one complete and rigid structure.

**Filter Frames.**—The filter cloths of leaching vats are supported on wooden frames, which are generally constructed in the following manner: A number of slats are laid on edge and parallel to each other, at about 6 to 9 in. centres, and on the top of these, at right angles to each, wooden strips are nailed, with 1-in. spaces between them, as shown in Fig. 74.

For wooden and masonry vats the usual dimensions are: Slats, 3 in.  $\times$  1 in., at 6-in. centres. Strips, 1 in.  $\times$  1 in. The quantity of timber in such a frame is exactly equal to the area covered, by 1 in. thick, or is equal to 0.0833 cu. ft. for each square foot of vat area.

For large steel vats the sizes shown in Fig. 74 are suitable. In this case the quantity of timber per square foot of vat is 1.272 sq. ft., reduced to

1 in. thick, or 0.106 cu. ft. The frame should be made in several separate sections, each section being braced by diagonals let in flush with the tops of the slats before the strips are nailed on, as shown on the left-hand side of Fig. 75. The ends of the slats and strips should be about 1 inch clear of the sides of the vat.

The filter cloth lies on the strips, and in wooden vats is sometimes nailed to the sides by  $\frac{3}{4}$ -in. clout nails. Otherwise curved strips of wood are fixed round the outside of the filter frame, and the edge of the cloth is tucked in and wedged with a rope gasket, between the strips and the side of the vat. Sometimes boards pierced with numerous holes are laid flat on the slats

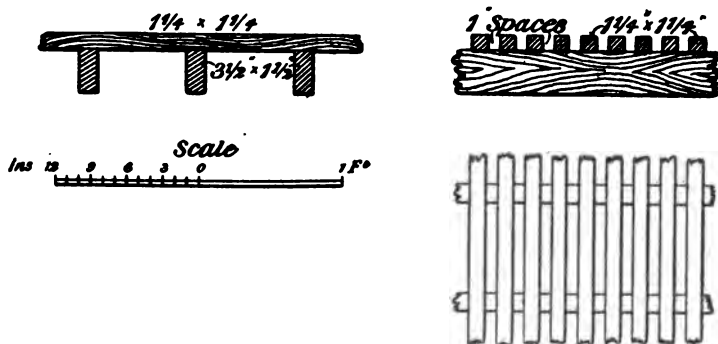


FIG. 74.—Filter Frames.

instead of the strips to support the cloth, but this is inferior to the foregoing method, because it considerably reduces the effective filtering area.

**Filter Cloths for Vats.**—Coconut fibre matting is the usual material for this purpose, and it may be used alone, or preferably with a cloth of hessian above it. The matting should be of the best quality obtainable, but as regards the hessian, a medium quality should be selected, as it filters better than the more expensive kinds. Canvas and duck have been occasionally tried, but cannot be recommended for this purpose.

The filter cloths are made up to a circle 10 to 12 in. larger than the vat, and when made in sections, these separate parts should overlap at least 6 in. when laid in place, this overlap being in addition to the extra diameter above mentioned. The outside edges are bound with strong canvas 6 in. wide, and the edges round discharge holes with 4-inch canvas. Instead of making the holes in the main filter cloth a tight fit round the discharge tubes, it is more convenient to use at each door a separate hessian mat, about 3 ft. 6 in. square, with a central hole in it, so cut and bound that it will spring tightly around the discharge tube, and so cover the larger hole in the main cloth.

**Shovelling Strips.**—To preserve the filter cloth from damage while shovelling out the residues, strips are lightly fixed above it, at right angles to the strips below. Pieces 2 in.  $\times$  1 in., laid flat, are sufficient for the purpose.

with 6-in. spaces between them, and they should be fixed with as few screws as possible, so as to be readily removed.

**Discharge Doors for Vats.**—Bottom discharge doors, attached to the lower

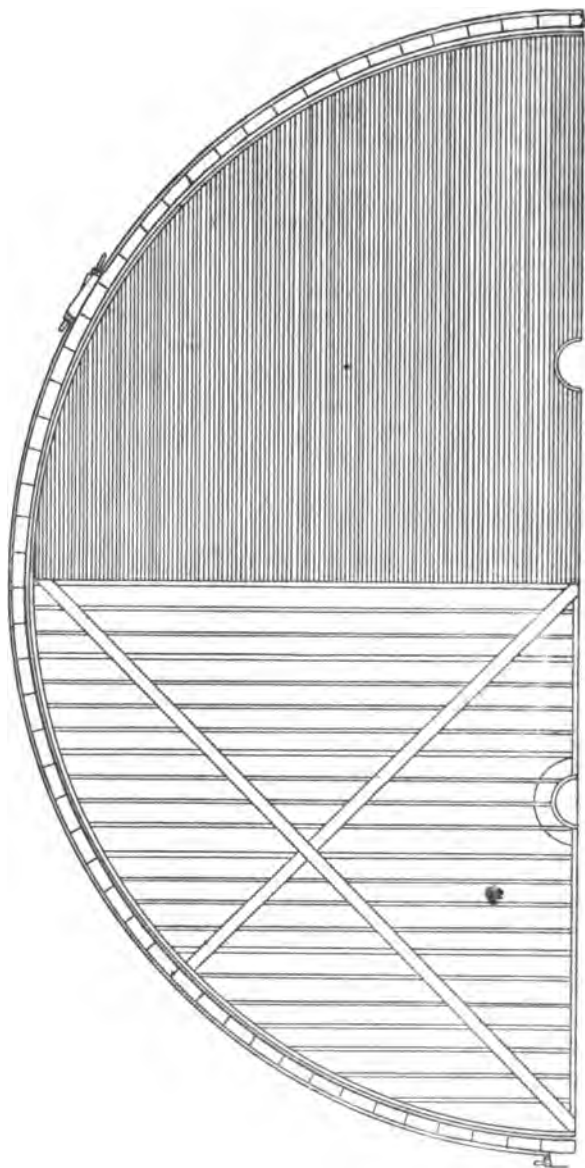


FIG. 75.—Filter Frame.

end of tubes passing through the filter cloth and frame, were introduced by Chas. Butters in 1891 in Johannesburg, and these are now used, sometimes with modifications in detail of construction, in all parts of the world. The side discharge door, whose lower edge is flush with the filter cloth, was introduced

by W. R. Feldtmann about the same time, but, for the reasons given below, the latter is not as good as the former. For instance, in a 20-ft. vat with one central bottom discharge, none of the material has to be shovelled more than 10 ft., and the average distance is 6 ft. 8 in., while, with a single side discharge in the same vat, some of the material has to be moved 20 ft., and even with two side discharges, diametrically opposite to each other, the maximum distance is still 14 ft. 2 in. But the diagrams Figs. 76 to 79 show more clearly than a written explanation the relative merits of side and bottom discharge.

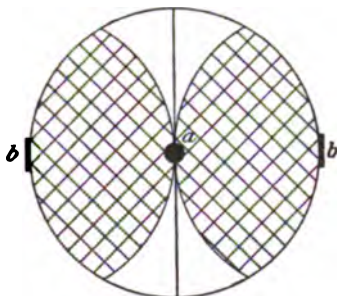


FIG. 76.—Discharging Residues.

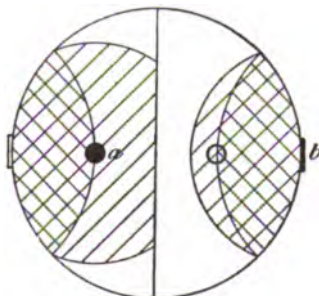


FIG. 77.—Discharging Residues.

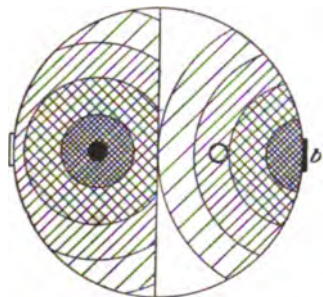


FIG. 78.—Discharging Residues.

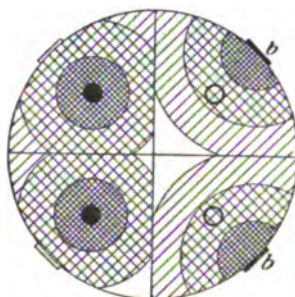


FIG. 79.—Discharging Residues.

Fig. 76 shows a vat with one central bottom discharge *a* and two side discharges *b b*. Fig. 77 has two of each kind. In each of these two diagrams the cross hatching indicates the portions of the vat contents which can be removed with equal labour by either system, that is to say, the portions which lie symmetrically between the opposite doors. Further, in diagram Fig. 77 the single-hatched portions show the additional amounts that can be discharged with the same maximum throw as for the cross-hatched portion; the left-hand side of the figure shows the extra amount for the bottom door, and the right hand of the figure for the side discharge door. The advantage of the first-named is apparent; and it is also evident that in both figures the bottom discharge door is much more favourably situated for the removal of the remaining unhatched portions.

Fig. 78 shows on the left hand for bottom doors, and on the right hand for



side doors, the amount of material that can be discharged by the two methods respectively, for the series of shovelling limits indicated by the various kinds of hatching. Fig. 79 is a similar diagram for two doors of each kind. In both these figures the darkest shading shows the material included within a radius of one-eighth of the vat diameter, the next zone has a radius equal to one-quarter of the diameter of the vat, and so on. The general result indicated is that, with a short throw, the bottom doors discharge twice as much as the side doors, and are therefore much more economical in practical work.

**Best Position for Discharge Doors.**—It is convenient to deliver the same quantity of material from each door, therefore the first rule is to divide the

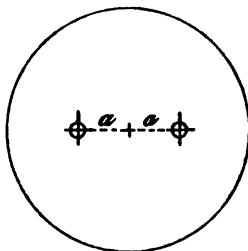


FIG. 80.—Vat with Two Discharge Doors.

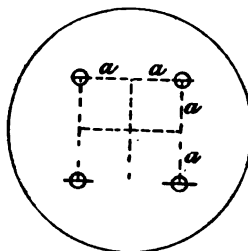


FIG. 81.—Vat with Four Doors.

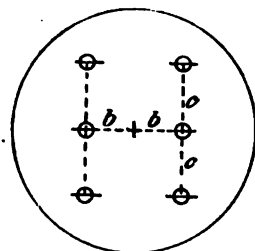


FIG. 82.—Vat with Six Doors.

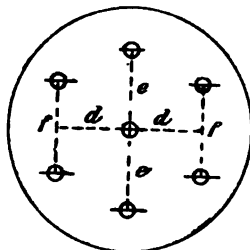


FIG. 83.—Vat with Seven Doors.

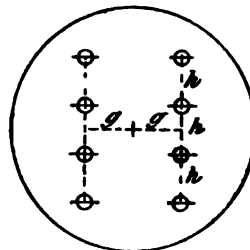


FIG. 84.—Vat with Eight Doors.

bottom of the vat into as many equal areas as the number of doors. The second rule is to place each door at the centre of gravity of its own section. Having done this, the positions have then to be altered sufficiently to bring the doors into a series of straight lines. This will then be the best result from theoretical and practical considerations combined, and the supporting structure of the vats should be laid out so that the discharging tram lines or conveyors can be placed along the lines so found. In Figs. 80 to 84 are shown suitable arrangements, for various numbers of doors.

For two doors (Fig. 80), the distance  $a = \text{radius of vat} \times \cdot 4244$ .

For four doors (Fig. 81),  $a$  also  $= \text{rad.} \times \cdot 4244$ .

For six doors (Fig. 82),  $b = \text{rad.} \times \cdot 4$  and  $c = \text{rad.} \times \cdot 55$ .

For seven doors (Fig. 83),  $d = \text{rad.} \times \cdot 60$ ,  $e = \text{rad.} \times \cdot 66$ , and  $f = \text{rad.} \times \cdot 75$ .

For eight doors (Fig. 84),  $g = \text{rad.} \times \cdot 4$  and  $h = \text{rad.} \times 4 \cdot 13$ .

**The Butters Discharge Door.**—Fig. 85 illustrates a 16-in. door of cast iron applied to a wooden vat. A circular hole, 16 in. in diameter, is cut

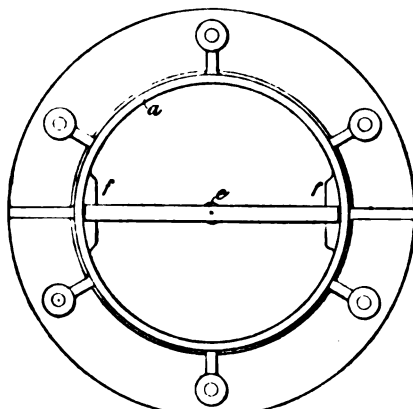
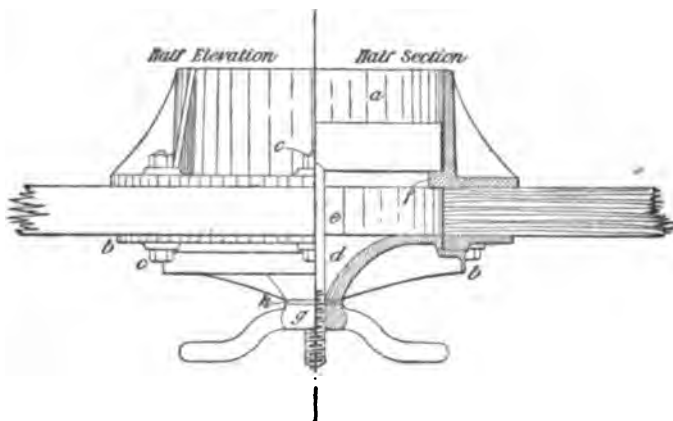
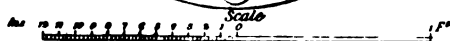
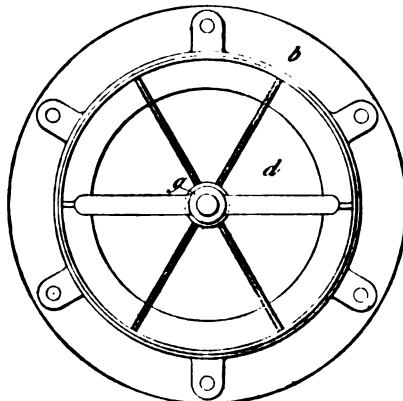
*Plan from above**Plan from below*

FIG. 85.—Butters' Discharge Door.

in the floor of the vat; a cast, flanged tube *a* is placed above this, a flange *b* below the opening, and the two are bolted together by six  $\frac{3}{4}$ -in. bolts *c* passing through the wood. The upper side of the door *d* and the lower side of the flange *b* are faced in the lathe to ensure a well-fitting joint. A tee bolt *e* rests on snugs *ff* which are cast in the tube *a*. This bolt passes through a central hole in the door, and the latter is screwed up tightly by a butterfly nut *g* on the tee bolt. A washer *h* is usually placed between the door and nut, and the bosses of both should be turned truly square with the axis of the bolt. When the door is screwed home, the tube is filled with sand and a slab of well-worked clay is pressed carefully over the top of the tube *a* to make a watertight joint. The tank is then ready for filling.



FIG. 86.—Discharging Tube.

When the residues are to be discharged, a truck is placed underneath and the door is taken off. Often-times it is sufficient to push a rod down through the sand to start it running through the door, but sometimes it is necessary to dig a hole down to the discharge tube. As it is somewhat dangerous to dig down more than 8 or 9 ft., an additional device has been used for vats of this depth and over, and this is shown in Fig. 86. A sheet of  $\frac{1}{8}$ -in. steel is bent to form a nearly complete tube *i*, leaving, however, a longitudinal slot *k* about 2 in. wide. It is bound and stiffened by two rings of flat, bar iron, *j j*, 2 in.  $\times$   $\frac{1}{4}$  in. The lower end of the tube fits into the tube of the discharge door, and if necessary the lower end of a second tube fits the top of the first, and so on. These tubes are fixed in place before the vat is filled, and consequently, for

discharging, it is only necessary to dig down in the first place till the uppermost tube is reached. These tubes are successively removed as the work of discharging proceeds.

A quick way of piercing a hole and starting the sand to run out is by means of a fine jet of water under pressure, applied at the top directly over the discharge door. A half-inch hose pipe will pierce 10 ft. in a few minutes by this means if a fairly high pressure be available.

A simple form of door has been used in S. Africa and America, which consists merely of a taper plug and a plate with a hole of corresponding taper attached to the vat. A long handle fixed to the plug projects through the top of the tailing, so that the plug may be withdrawn when required. It is not so safe, as regards leakage, as the more usual forms, and it sometimes happens that the plug sticks, and is not readily removed when required.

**Hinged Doors.**—Fig. 87 illustrates a convenient form of discharge door made, by the United Engineering Co. of Johannesburg, almost entirely of pressed steel. It consists essentially of a flanged tube made in two parts *a* and *b*, a door *c*, and a lever *d*, which is pivoted at its centre to the door, hinged

to the bracket *e* at one end, and supported at its other end by a swinging bolt *f*. The details of its construction can be understood from the figure

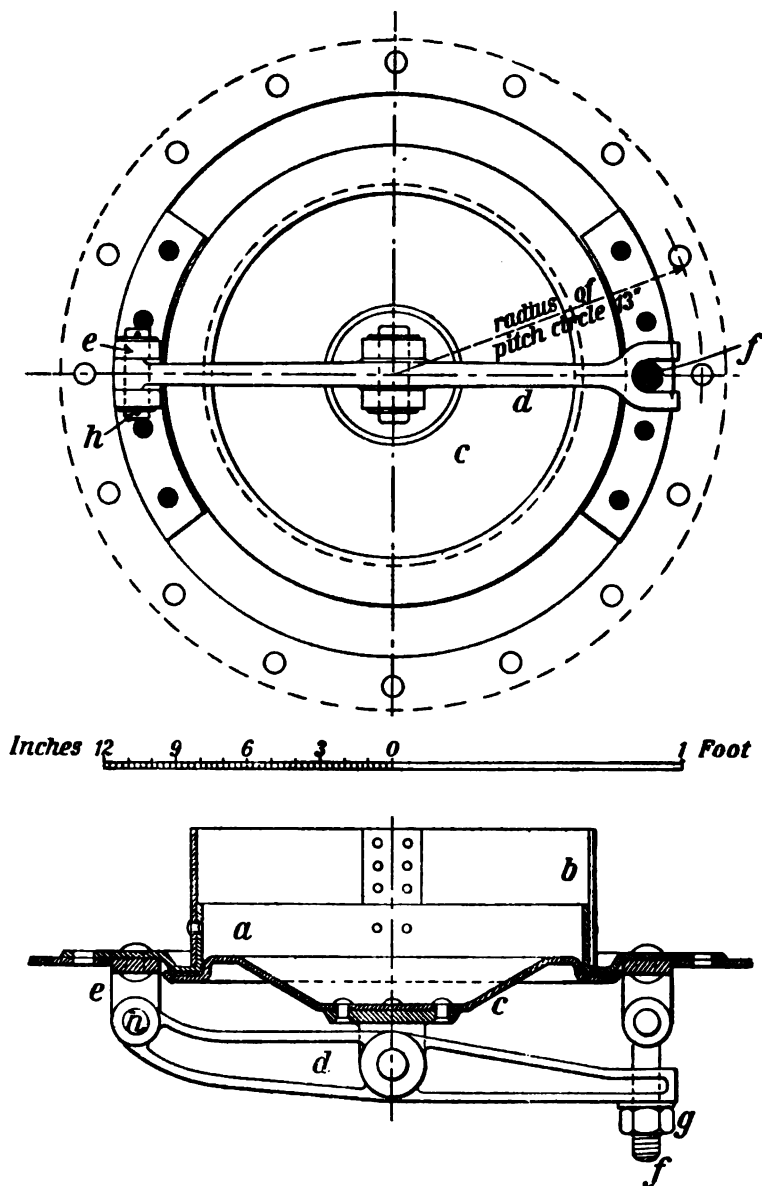


FIG. 87.—Hinged Discharge Door; Section and Plan.

without further description. For discharging, the nut *g* is slackened, the bolt *f* is swung to the right, and the lever and door are swung to the left on the pin *h*. The door may be left hanging downwards, or it may be hooked back

on to the bottom of the vat, out of the way of the workmen in the tunnel. It may be noted that the leverage obtained when tightening is twice as great as it is with the central bolt in Fig. 85, and by dispensing with the bolt hole in the door one possible source of leakage is avoided.

With either form of door a gasket of rubber or of asbestos cord can be used to make good the joint between door and flange.

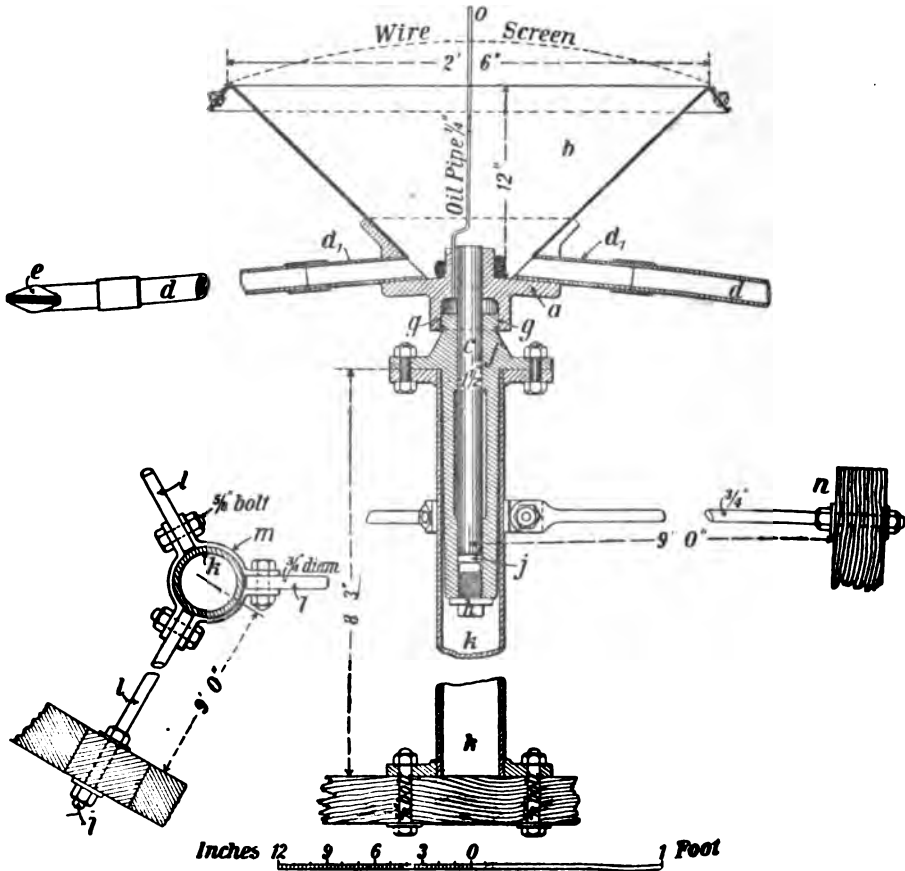


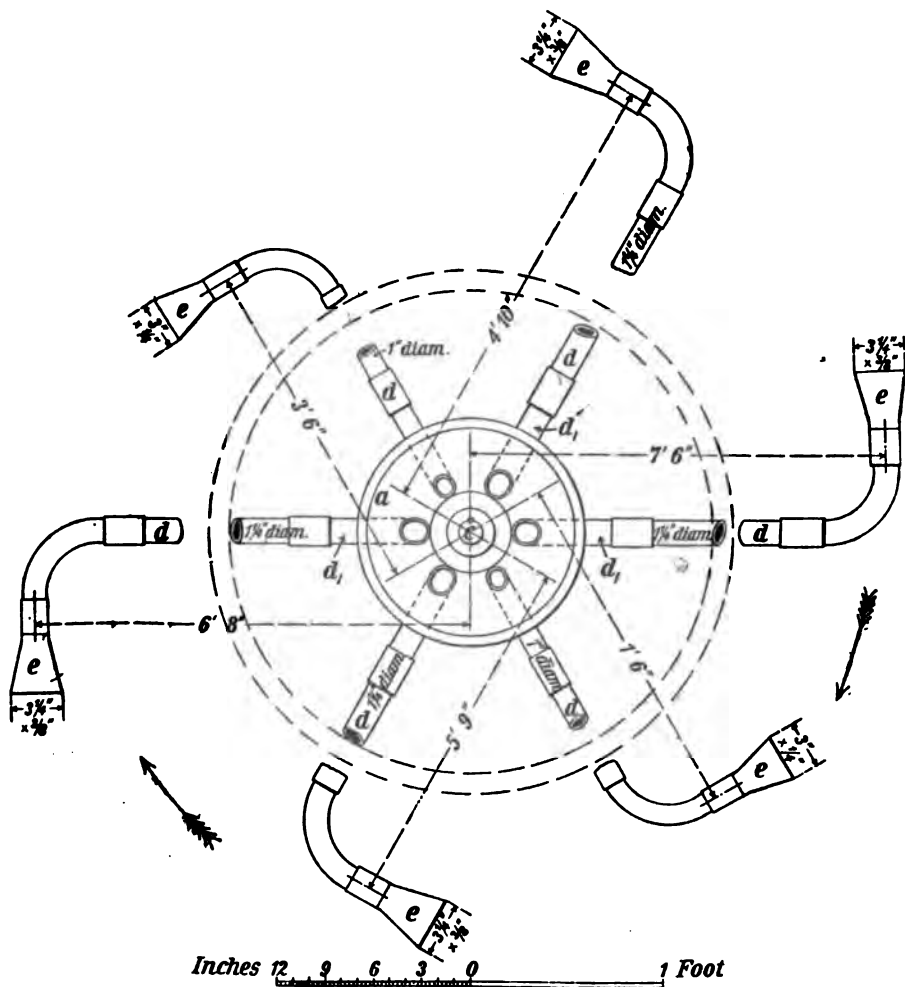
FIG. 88.—Butters-Mein Automatic Distributor in sectional elevation.

FIG. 89.—Plan of connection with staves.

Square doors, hinged on one side, are made by the Pacific Tank Co. of San Francisco. In these a removable bridge bar is attached at both ends by bolts to the door frame, and carries a central set screw with a wheel handle.

**Distributors for Collecting-Vats.**—The best automatic apparatus for delivering battery pulp to collecting vats is the distributor devised by Chas. Butters and the late G. A. Mein, which is illustrated in Figs. 88 to 90. Fig. 89 shows the method of staying the central pipe which carries the apparatus.

The movable part consists of a casting *a*, carrying a sheet-iron hopper *b*. This casting is keyed to a vertical steel shaft *c*, and is provided with six radial pipes *d d*, attached to it in the following manner: short lengths *d*<sub>1</sub> of these pipes, having both ends screwed, are laid in the mould in the foundry and



**FIG. 90.—Butters-Mein Automatic Distributor in Plan.**

cast in, thus avoiding the cost of boring and tapping the holes in the casting *a*. A flat nozzle *e*, formed of sheet iron and brazed to a pipe socket, is attached by a bend, and a suitable length of ordinary black pipe and another socket to each of the short pipes *d*<sub>1</sub> in the casting *a*. The latter is bored out for the spindle and also to fit the lower casting *f* at the part marked *g*. The lower casting *f* is bored to form a bearing for the shaft *c*, and also turned to fit the upper casting *a*, the idea being to exclude all grit from the bearing.

At its lower end a steel plug *h* is screwed in to serve as a footstep for the shaft. The plug is turned slightly convex on its upper end, and one or more turned steel washers *j* are placed between the shaft and plug. The plug and washers should each be hardened and tempered.

A pipe *k* with a flange at each end carries the whole apparatus, and is stayed by three rods *l l l* clamped on to it, as shown at *m* in Fig. 89. The outer ends of these stay rods pass through the staves as seen at *n*, and are held by nuts and washers on either side of the timber.

An oil pipe *o* is provided for lubricating the bearing and footstep.

When the pulp enters the hopper *b* it passes out by the radial pipes *d d* and the nozzles *e*, and the reaction of the issuing water against the bends at the outer ends of the pipes *d d* is sufficiently powerful to cause these pipes to revolve in the direction shown by the arrows. In consequence of this rotation, each pipe delivers a stream of pulp over an annular space in the vat, and the pipes must be so adjusted, as to their relative lengths and diameters, that the supply of pulp is as nearly as possible equal on every unit of surface of the vat. For instance, if the pipes are of equal diameter, and the inclination so arranged as to give an equal discharge from each, then the area of the vat should be divided by circles into twice as many equal areas as there are pipes. The length of each pipe is then determined by the fact that it feeds two of the areas so obtained, and that, therefore, its nozzle must be over the circle which divides these two areas. If the radial pipes have different diameters, the annular spaces must be proportioned accordingly, and the most uniform results are obtained by having the longer pipes of larger diameter than the short ones. Also, it is generally advisable to make the longest pipe less than the length determined as above, so that it shall not deliver too close to the outer edge of the vat; and to achieve this result, the best plan is to design the pipes strictly according to the principles laid down, but for a vat 1 ft. less in diameter than that in which they are to be actually used.

A wire screen with about three holes to the linear inch should be attached to the flange of the hopper by six small bolts and nuts, to prevent choking of the radial pipes.

It has been found experimentally that the actual pulp delivery by a distributor constructed as shown in the drawings is about 59 per cent. of the theoretical flow of water, calculated from the available head and resistance of pipes.

## CHAPTER XXXVI.

### PIPING, COCKS, AND LAUNDERS.

**Pipes for Leaching Plants.**—These may conveniently be considered under three heads:

- (a) The delivery of solution from storage vats to leaching vats;
- (b) The conveyance of the solution from leaching vats to the precipitation boxes; and
- (c) The return of solutions to the storage vats.

In all three cases the greatest attainable simplicity is desirable, in order that the arrangement of pipes can be readily understood and remembered by the man on shift. Also, for the same reason, the number of cocks should be as few as possible, and every cock should be placed in a visible and easily accessible position. These apparent truisms are often neglected in the design of works.

(a) For delivering the various solutions to the leaching vats, it is well to have large pipes, so that the vats may be quickly filled; and suitable sizes for this purpose will be found in Table LVI. The simplest and therefore best arrangements are shown in Figs. 91, 92, and 93. In each figure the storage vats are marked *SS*, and the leaching vats *LL*, while cocks are indicated by small black circles.

In Fig. 91, where the storage vats are assumed to be at a higher level than the leaching vats, each of the former has an outlet pipe which leads to all of the latter, and is provided with a branch outlet and cock over each of the leaching vats. Cocks are also placed at *aa*, but these are only necessary when the pipes have to be disconnected for any purpose. They are otherwise superfluous, so that in ordinary working they would remain always open, and may be neglected, in comparing this arrangement with other systems of piping, in respect of simplicity of operation. Under these circumstances, it will be seen that the minimum number of cocks required is equal to the number of leaching vats multiplied by the number of storage vats.

Or if *N* equals the number of leaching vats, and *n* the number of storage vats, the number of cocks will be *Nn*, i.e. 12, in Fig. 91. It is a good plan to paint these pipes different colours, such as red for strong solution, yellow for weak, and blue for alkaline washes. It is not necessary to paint them throughout their length, but only at the delivery cocks, so that the shiftman should make no mistake. This system of piping was at one time almost universally adopted, as it has the advantage that all the solutions can be supplied at the same time to any of the treatment tanks.



In the next system, shown in Fig. 92, one large pipe only is used instead of several smaller ones. Thus the solutions can be applied so quickly that the vats can be served in turn by the one pipe for all the different washes. This arrangement requires a little more care and attention on the part of the workman, because at each operation he has to open one cock at *b* as well as one at *c*, and also has to see that all the other cocks are closed, especially the other *b* cocks.

It has, however, the merit of great simplicity, and by using the same notation as before, the total number of cocks required is only  $N + n$ , i.e. 7, as in Fig. 92. This advantage is more apparent in plants with many vats;

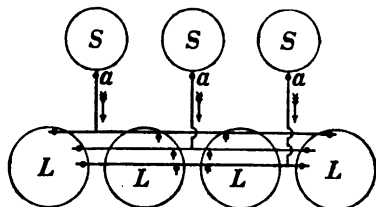


FIG. 91.—Arrangement of Pipes and Cocks.

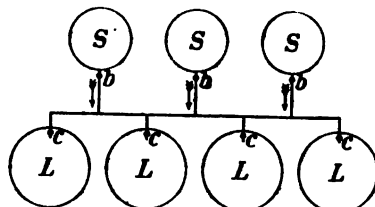


FIG. 92.—Arrangement of Pipes and Cocks.

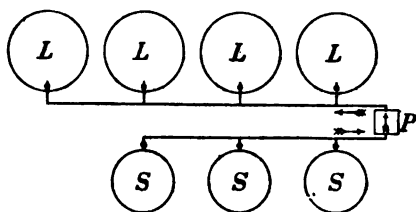


FIG. 93.—Arrangement of Pump, Pipes, and Cocks.

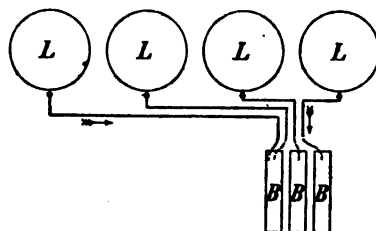


FIG. 94.—Arrangement of Leaching Pipes.

thus, with four storage and twelve leaching vats, the first system described above needs forty-eight cocks, and the second only sixteen.

The arrangement shown in Fig. 93 is usually adopted when the solutions have to be pumped to the leaching vats. It is essentially the same as that last described, although it appears slightly different owing to the introduction of the pump *P*. It is important to note that, in this case, one of the delivery cocks must be opened before the pump is started, and it is a good rule to always leave open the last cock used until the next one required has been opened.

(*b*) For leaching purposes, each vat should have a separate pipe, with a separate visible outlet, because the flow of solution from the vat affords the only evidence as to whether percolation is proceeding in a satisfactory manner. The best plan is to provide a pipe from each vat to the precipitation shed, as in Fig. 94. These pipes may terminate close to the precipitation boxes *B B B*, and each may be provided at the end with a piece of flexible hose, long enough to deliver into either of the boxes, as indicated by the dotted lines.

For vacuum leaching a closed system of pipes is necessary. Each leaching pipe has a cock near its outlet, and a branch also containing a cock, connecting to a common pipe which leads to the vacuum chamber.

(c) **Returning Solution to Storage Vats.**—When these are below the precipitation boxes, Fig. 95 shows the ordinary system of piping adopted when each box is used for one kind of solution only. No cocks are then required ; but if more than one kind of solution is passed at different times through the

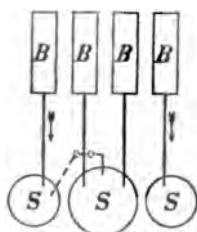


FIG. 95.—Return of Solution to Storage Vats.

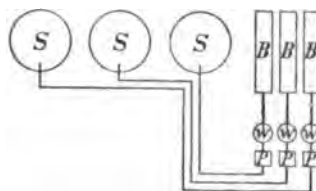


FIG. 96.—Return of Solution to Storage Vats.

same box, its outlet pipe must be provided with branches and cocks for the corresponding storage vats as shown by the dotted lines.

When the storage vats are above the boxes, the arrangement shown in Fig. 96 is suitable. In this figure, boxes and vats are lettered as before, *ww* represent small sumps, and *PP* are pumps. In this case also no cocks are required unless, as before, different solutions are passed through the same box, when the same modification is necessary.

Table LVI. gives the sizes of pipes which have been found suitable for the various purposes under the conditions above described.

TABLE LVI.—*Sizes of Solution Pipes.*

| System of Piping.                               | Diameter of Vats. |                |                |                |                |                |
|---|-------------------|----------------|----------------|----------------|----------------|----------------|
|   | Under 20 feet.    | 20 to 26 feet. | 26 to 32 feet. | 32 to 40 feet. | 40 to 50 feet. | 50 to 60 feet. |
| Charging vats by method shown in Fig. 91        | 2 in.             | 2½ in.         | 2½ in.         | 3 in.          | 4 in.          | 5 in.          |
| Charging by method of Figs. 92 and 93           | 2½ „              | 3 „            | 4 „            | 5 „            | 6 „            | 8 „            |
| Leaching from treatment vats as in Fig. 94      | 1 „               | 1½ „           | 1½ „           | 2 „            | 2½ „           | 3 „            |
| Returning to storage vats as in Figs. 95 and 96 | 1½ „              | 1½ „           | 2 „            | 2½ „           | 3 „            | 4 „            |

**Piping for Decantation Slime Plants.**—For this purpose such large sizes are necessary that launders should be substituted as far as possible, a practice which also assists the aeration of the pulp and solutions.

The principal requirements are—

- (a) Decantation pipes.
- (b) Pulp transfer.
- (c) Residue discharge.
- (d) Solution pipes.

- (a) If the decanted liquor can gravitate from the vat, it is only necessary to provide a short outlet pipe discharging into a launder as at *a*, Fig. 97. This, however, is seldom done, as it is usual to connect all the decanter outlets, each provided with a valve, to a main pipe which conveys the decanted solution to the extractor house.

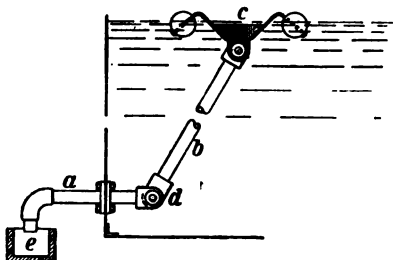


FIG. 97.—Decanting Apparatus.

The decanter itself is a pipe jointed at its lower end, so that it can turn freely in a vertical plane. It has, at the top, a slotted cross pipe or an enlarged mouth

having a large periphery, so that the clear solution can overflow into the decanter at a low velocity.

Fig. 97A illustrates a simple method of making the hinged joint.

- (b) For pulp transfer a pipe system as shown in Fig. 98 is used. There is a network of pipes *a a*, shown in dotted lines, connected from the bottom

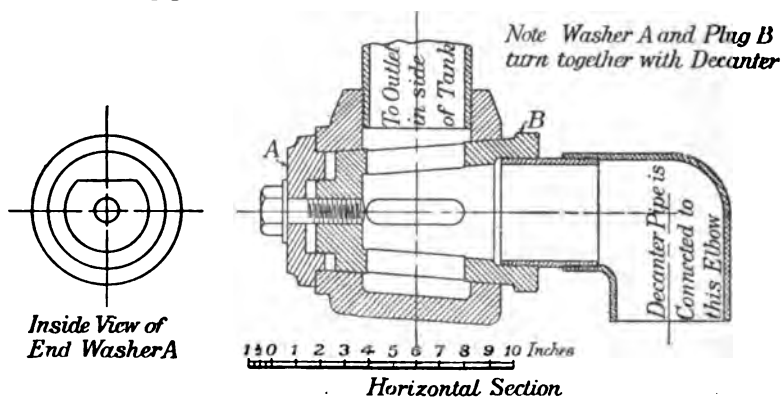


FIG. 97A.

of each vat to the suction of the pumps *P P*, which deliver through the pipes *b b*, shown in full lines, to the top of any vat. Each branch of *a* and *b* pipes has a valve. By this means the contents of any vat can be circulated through the same vat, or transferred to any one of the others. The valves are indicated by crosses. The pipes *a a* are brought through the trenches or tunnels under the vats which give access to the discharge valves. The pipes *b b* are usually slung under the working platforms.

- (c) Where slime residue has to be lifted, the arrangement in Fig. 98 will also serve for this purpose, by providing an additional outlet and valve

to the pipes *b b*. When the residue can gravitate to the slime dams, the pipes *a a* instead of *b b* may be provided with an extra outlet and valve.

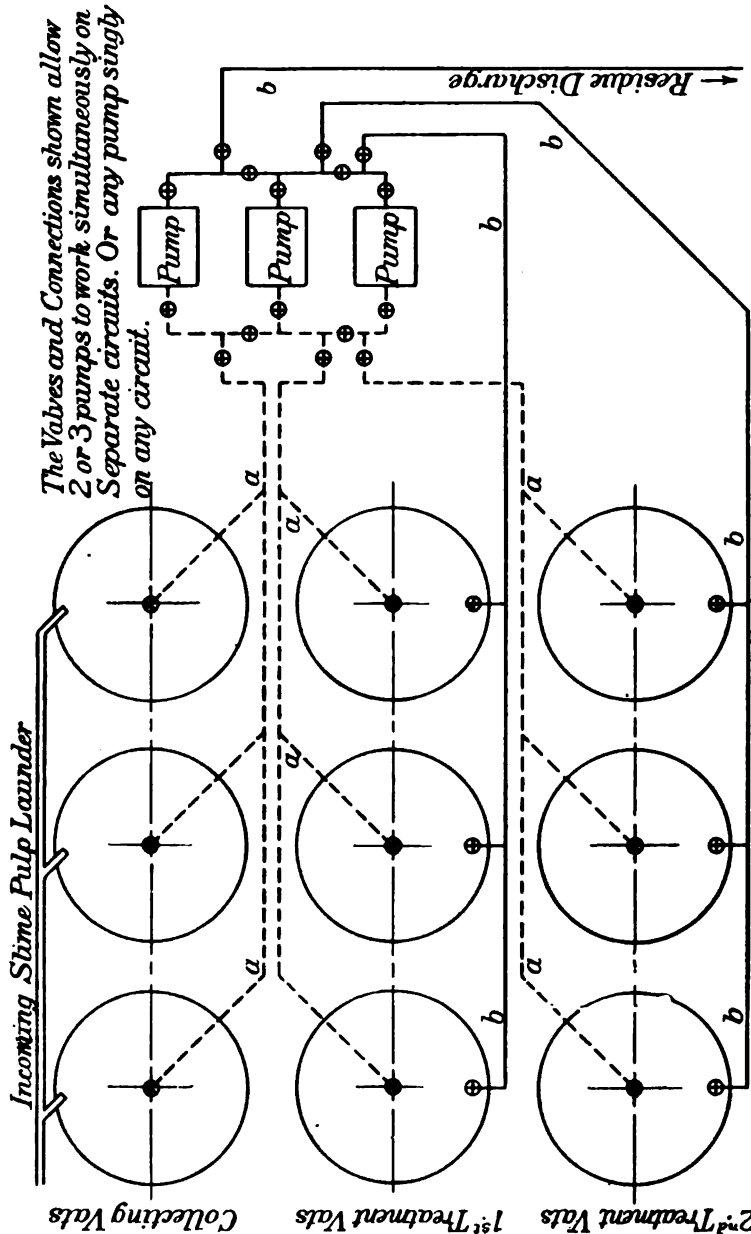


Fig. 98.—Diagram of Studge Piping for Decantation Plant.

(d) In decantation plants the solution piping consists of a main and branches led by the most convenient route to serve every vat. This route is generally along the platforms placed between the rows of vats. The

vertical centre pipe with rubber-disc valve, as shown in the drawings of the Robinson slime plant, is omitted in more recent plants, a gate valve immediately under the outlet being now used. As only one vat at a time receives solution, all the mains and branches may be of the same bore throughout the system. Suitable sizes for the several services, derived from current practice, are given in Table LVII.

TABLE LVII.—*Sizes of Pipes for Decantation Slime Plants.*

| Purpose   | Diameter of Vats. |                |                |                |                |                |
|---|-------------------|----------------|----------------|----------------|----------------|----------------|
|   | Under 20 feet.    | 20 to 30 feet. | 30 to 40 feet. | 40 to 50 feet. | 50 to 60 feet. | 60 to 70 feet. |
| (a) Decantation pipes to extractor house, .     | 2½ in.            | 3 in.          | 3½ in.         | 4 in.          | 5 in.          | 6 in.          |
| (b) and (c) Pulp transfer and residue discharge | 3 to 4 in.        | 4 to 6 in.     | 5 to 8 in.     | 6 to 10 in.    | 8 to 12 in.    | 9 to 15 in.    |
| (d) Solution supply, .                          | 2½ in.            | 3 in.          | 4 in.          | 5 in.          | 6 in.          | 7 in.          |
| Sluicing hose . .                               | 1½ „              | 2 „            | 3 „            | 3½ „           | 4 „            | 4½ „           |

**Piping for Agitation Vats.**—In agitation treatment the processes are more continuous than in the case of decantation, and the pipes are therefore smaller. Thus for Pachuca vats the following sizes are used to pass the pulp from one vat to another :

|                            |             |             |             |
|----------------------------|-------------|-------------|-------------|
| Size of vat . . . . .      | 30 × 10 ft. | 36 × 12 ft. | 45 × 15 ft. |
| Diameter of pipe . . . . . | 4 in.       | 5 in.       | 6 in.       |

These pipes are provided with bypasses and valves to cut out any vat at will (see Fig. 99). They receive the pulp from the outlet of an overflow box at the top of each air lift, and discharge through the side of the next vat. For other piping the general rules given below may be used.

**General Rules for Pipes.**—Pump connections for water or solution usually have the same diameter as the suction inlet and delivery outlet, respectively, of the pumps. The velocity of flow should not exceed 6 ft. per second with reciprocating pumps, or 8 ft. per second for centrifugals. If the velocity of pump suction or delivery exceeds these figures, larger pipes should be used and connected to the pipe flanges by reducers.

For pipes under gravity heads the Eytelwein formula is suitable :

Let  $d$  = diameter of pipe required.

$h$  = head in feet.

$L$  = length of pipe in feet.

$Q$  = cubic feet of water discharged per minute.

then :

$$d = 0.538 \sqrt[5]{\frac{LQ^2}{h}} \quad (1)$$

**Cases of very low Head.**—In special instances, such as the discharge pipe from a rim launder, when the head above the opening of the pipe is very

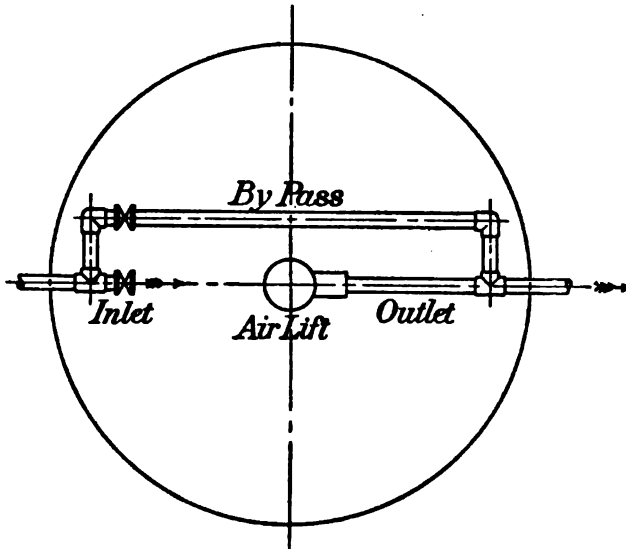


FIG. 99.—Arrangement of Bypass Piping for Pachuca Vat.

small, it is necessary to calculate the size of the hole in the launder. This may be done in the following manner :

Let  $h$  = head, in feet, above the hole.

$Q$  = quantity of liquid in cubic feet per minute.

$d$  = diameter of hole in inches.

then

$$d = \frac{\sqrt{Q}}{1.28 \sqrt{h}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

The size of the pipe having also been determined, it should be connected to the larger hole of the rim launder by a short length of pipe of gradually diminishing cross section. The depth  $l$  of this must be such that the total head  $h$  above its lower end is sufficient to give the velocity chosen for the pipe.

If  $v$  = actual velocity in feet per second,

then 
$$h = \frac{v^2}{25} \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

This formula is based on the assumption that the actual flow is 63 per cent. of the theoretical. This is sufficiently accurate for the purpose in view.

**Fixing Pipes.**—It is convenient when laying pipes to use crosses for connections at right-angled turns, as at  $c$  in Fig. 100, instead of the usual bend shown at  $b$  in the same figure. The benefit arising from this is that, by removing either of the plugs  $d$ , the corresponding pipe can be readily cleaned

and obstructions removed. This practical advantage far more than counterbalances any frictional losses which may result from the substitution of the cross for the bend. Crosses can often be used in place of tees for the same purpose. It may be remarked, incidentally, that all pipes and fittings in any one plant should be of one make only, as unfortunately the British makers have not yet adopted standard sizes and threads.

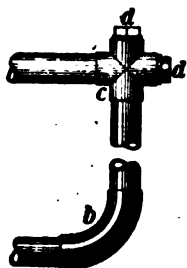


FIG. 100.—Fixing Pipe Connections.

Red-lead joints are unsuitable for permanent plants, because the action of cyanide causes leakage after a time. Paraffin wax melted in a little cylinder oil makes satisfactory joints for cyanide solution pipes. The portion of the piping which must be fixed with the greatest precaution is between leaching vats and the precipitation boxes, because it is in this part of the circulating system that the solution carries the maximum amount of gold.

All pipes laid below ground level should be placed in  $1\frac{1}{2}$ -in. wooden troughs, with removable covers held by screws, the top of the covers being preferably level with the ground.

When pipes have to be carried overhead in the open, the device shown in Fig. 101 forms a neat and effective support. In this figure, *a* is a  $2\frac{1}{2}$ -in. solution pipe resting on one-half *b* of a 3-in. cross which has been cut into two pieces, as shown in Fig. 101, and which therefore serves for two standards. The half cross is screwed on to the pipe *c*, which is attached at its lower end to a base plate *d* by the flange *e*.

Neat brackets for supporting pipes from sheds or tanks may be made from bends, flanges, and split crosses in a similar manner.

Table LVIII. is given to enable the spacing of supports for pipes to be readily determined, as also the load on the supports.

**Cocks.**—The ordinary plug cocks are often used in cyanide plants, but are not at all suitable, as they soon wear when grit finds its way between the surfaces, and leak. Where employed, the plugs should be removed regularly, say once a week, and smeared with vaseline or other thick mineral lubricant, and when they show any signs of wear they should be refitted. This is often inconvenient, on account of the position of the cock, and for this reason plug cocks are seldom used for pipes over 3 in. in diameter. For larger pipes one of the numerous types of gate valve, with packed spindle, is generally used.

The packing may be of asbestos, but a cheaper and effective material is hemp steeped in molten paraffin wax, thinned with a little cylinder oil. For rich gold solutions, the more expensive cocks and valves are to be recommended, but should be ordered with all parts made of iron or steel.

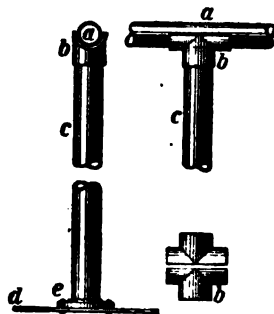


FIG. 101.—Pipe Carriers.

TABLE LVIII.—*Weight of Solution Pipes and their Contents and Limiting Spans.*

| Nominal Diameter in inches. | Weight per foot, lb. | Weight of Solution in lb. per foot at 64 lb. per cubic foot. | Total Weight full, per foot run. | Approximate External Diameter in inches. | Maximum Distance between Supports consistent with Limiting Deflection of $\frac{1}{16}$ in. |
|-----------------------------|----------------------|--|----------------------------------|--|---|
| 1 inch                      | 1.91                 | 0.35   | 2.26                             | 1.31                                     | 8 feet 0 inches   |
| 1½ "                        | 2.73                 | 0.55   | 3.28                             | 1.62                                     | 9 " 6 "   |
| 1¾ "                        | 3.20                 | 0.79   | 3.99                             | 1.92                                     | 11 " 0 "  |
| 2 inches                    | 4.40                 | 1.40   | 5.80                             | 2.45                                     | 13 " 0 "  |
| 2½ "                        | 5.70                 | 2.18   | 7.88                             | 2.96                                     | 14 " 6 "  |
| 3 "                         | 6.84                 | 3.14   | 9.98                             | 3.47                                     | 16 " 0 "  |
| 4 "                         | 9.63                 | 5.59   | 15.22                            | 4.50                                     | 18 " 6 "  |
| 5 "                         | 11.87                | 8.73   | 20.60                            | 5.50                                     | 20 " 9 "  |
| 6 "                         | 15.40                | 12.57  | 27.97                            | 6.50                                     | 22 " 8 "  |

The Butterfield inverted plug-cock is suitable for cyanide plants, the design being illustrated in Fig. 102. Its special features are as follows:

(1) The tendency when opening and closing is to slacken the plug from its seat, against the pressure of a light spring *a*, which otherwise keeps the plug always up to its facing. The thrust of the spring is taken up on a ball bearing *b*. For these two reasons the cock works easily.

(2) Should the cock get foul with slime or grit, it can be cleaned without taking it apart, by pressing the plug downwards against the spring, thus allowing the liquid to flush out the body of the cock.

(3) The cock can be packed under pressure, and whether open or closed.

An adaptation of the laboratory rubber-tube clamp has been introduced to serve a number of purposes, in place of the usual metal cocks and valves. This, as shown in Fig. 103, consists of a short length of rubber and canvas hose, inserted in the piping at a convenient place and secured at both ends with clips A. B B are bars of hard wood which are drawn together by two bolts to regulate or stop the flow. The arrangement is particularly suitable for sand and slime pulp, and is said to be not only more durable than metal, but it also causes less annoyance from leakage. The rubber hose perishes

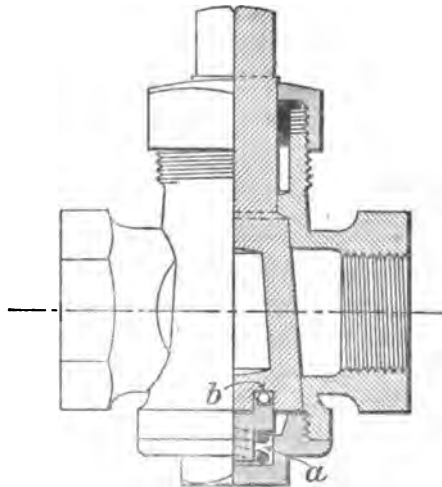


FIG. 102.—Butterfield's Patent Inverted Plug-Cock.



in time, but it is quickly replaced at a small cost. A number of these clamps are in use on pipes up to 5 in. in diameter.

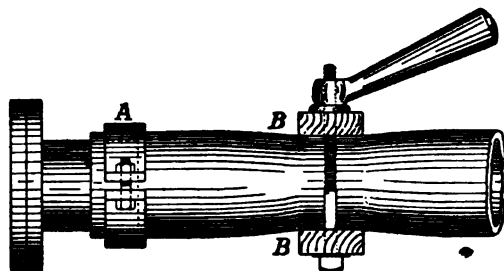


FIG. 103.—Bunsen Valve.

**Launders.**—As the relation between the width and depth of launders is generally a matter of guesswork, it may be worth while to show how to estimate a proper ratio between these two dimensions.

For launders of rectangular cross section : If  $W$  be the width of launder and  $D$  its depth, the cost is roughly proportional to  $W + 2D$ .

The quantity of liquid passing when running full is proportional to

$$WD \times \sqrt{\frac{WD}{W + 2D}},$$

and the greatest economy is secured when

$$\frac{\text{Cost}}{\text{Capacity}} = \frac{W + 2D}{WD} \sqrt{\frac{W + 2D}{WD}} = \left( \frac{W + 2D}{WD} \right)^{\frac{3}{2}}$$

is a minimum, that is, when  $W = 2D$ .

The maximum discharge is also obtained when width of flow equals twice the depth of flow. With launders running full, both conditions could be secured simultaneously, but in practice the depth of flow should not exceed two-thirds of the depth of launder, to allow for the inevitable pulsations which occur even with a fairly uniform flow, as, for instance, from a mill, a tailing wheel, or a pump. Where a badly designed air lift is used, the flow of pulp is often irregular, and the cross section of the launder should be about twice normal. In both these cases maximum economy of construction is obtained by the shape stated above, viz.  $W = 2D$ . There are, however, two reasons for departing from strict economy : (1) The least surface is subjected to abrasion as well as the greatest discharge is obtained when the depth of flow is half the width ; (2) to provide for pulsations, the actual margin of depth is probably of more account than a mere excess of sectional area.

The following rules are therefore adopted : In all cases calculate the area required for normal flow on the basis of depth of flow being equal to half the width. Then, for steady flow, make the depth of launder 50 per cent. greater than the depth of flow ; and, where large fluctuations are expected, 100 per cent. greater. Two standard proportions are, therefore, obtained.

In the one the launder depth is three-quarters of the width ; and in the other the width and depth are equal. Formulæ and tables suitable for these rules are given below.

**Calculation of Size.**—For the sizes usual in connection with cyanide plants, the ordinary formula with a fixed coefficient  $c$  will serve to calculate the velocity of flow and quantity delivered for any given or assumed dimensions, thus :

Let  $v$  = velocity in feet per second.

$R$  = mean hydraulic depth =  $\frac{\text{cross-section of flow}}{\text{wetted perimeter}}$ .

$S$  = sine of slope,  
= grade expressed as a fraction or percentage grade divided by 100.

$Q$  = cubic feet per second.

$$\text{Then } v = c\sqrt{RS} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

$$\text{and } Q = v \times \text{cross section of flow} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (5)$$

By actual measurements on launders, the following factors are given for the coefficient  $c$  :

For slime overflow from collecting-tanks,  $c = 85$ .

For average battery pulp (9 to 1),  $c = 75$ .

As very great accuracy is neither obtainable nor necessary, it will be sufficient in most cases to use the mean value, namely, 80, which agrees closely with that obtained by Kutter's formula for roughly planed timber or cement channels.

In using these formulæ, the size required is found by assuming certain dimensions, calculating the capacity thereof, and then trying larger or smaller sizes until suitable dimensions are found.

But where the rule of making width of flow equal to a fixed multiple of its depth is adopted, the formula can be so modified as to give the required size at one operation. Thus, for width equal to twice the depth of flow :

Let  $x$  = depth of flow in feet,

Then  $2x$  = width.  $2x^2$  = section of flow.

$4x$  = wetted perimeter and consequently the mean hydraulic depth  $R = \frac{x}{2}$ .

$$\text{Then } x = \sqrt[5]{\frac{Q^2}{2c^2S}} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (6)$$

which can be used for any value of  $c$ .

Or putting  $c = 80$  and simplifying

$$x = 0.1513 \sqrt[5]{\frac{Q^2}{S}} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (7)$$

Table LIX. is given to facilitate the use of formula (3). When the quantity  $\frac{Q^2}{2c^2S}$  is found, select the number nearest to it in the third column. Then the corresponding value of  $x$  = depth of flow in inches or feet in first and second columns respectively.

TABLE LIX.

| Depth of flow<br>in inches from<br>equation (3) or radius<br>in inches from<br>equation (5). | Fifth Roots of<br>Numbers in<br>Third Column.<br>Depth of Flow<br>in feet and<br>decimals. | Fifth Powers of Numbers in<br>Column 2. The numbers below<br>correspond to Values of $\frac{Q^5}{2c^5S}$ in<br>equation (6) or to Values of<br>$\frac{0.81 Q^5}{c^5S}$ in equation (8). |
|--|--|---|
| 3 inches   | 0.25   | 0.00098   |
| 3½ "   | 0.29   | 0.00205   |
| 4 "  | 0.33   | 0.00391   |
| 4½ "   | 0.37   | 0.00693   |
| 5 "  | 0.42   | 0.01307   |
| 5½ "   | 0.46   | 0.02060   |
| 6 "  | 0.50   | 0.03125   |
| 7 "  | 0.58   | 0.06564   |
| 8 "  | 0.67   | 0.13501   |
| 9 "  | 0.75   | 0.23730   |
| 10 "   | 0.83   | 0.39390   |
| 12 "   | 1.00   | 1.00000   |
| 14 "   | 1.17   | 2.16139   |
| 16 "   | 1.33   | 4.21399   |
| 18 "   | 1.50   | 7.59375   |
| 20 "   | 1.67   | 12.86008  |
| 24 "   | 2.00   | 32.00000  |

As an example, in the case of a 100-stamp mill crushing 5 tons of ore per stamp per day, with 9 tons of water per ton of ore, with a 3½ per cent. grade in the launder, the quantity  $Q$  is 1.736 cu. ft. per second.

TABLE LX.

| Depth of Flow<br>in inches. | Values of<br>$0.1513 \sqrt[5]{\frac{Q^5}{S}}$ ,<br>i.e. Depths of Flow<br>in feet. | Values of $\frac{Q^5}{S}$ in<br>equation (7). |
|-----------------------------|--|---|
| 3                           | 0.25   | 12.32   |
| 3½                          | 0.29   | 25.86   |
| 4                           | 0.33   | 49.40   |
| 4½                          | 0.37   | 87.44   |
| 5                           | 0.42   | 164.80  |
| 5½                          | 0.46   | 259.72  |
| 6                           | 0.50   | 394.07  |
| 7                           | 0.58   | 827.70  |
| 8                           | 0.67   | 1703  |
| 9                           | 0.75   | 2992  |
| 10                          | 0.83   | 4967  |
| 12                          | 1.00   | 12610   |
| 14                          | 1.17   | 27647   |
| 16                          | 1.33   | 52473   |
| 18                          | 1.50   | 95760   |
| 20                          | 1.67   | 163819  |
| 22                          | 1.83   | 258841  |
| 24                          | 2.00   | 403531  |

Taking  $c$  at 75—

$$r = \sqrt[5]{\frac{3.014}{2 \cdot 5825 \cdot .035}} = \sqrt[5]{.00765}.$$

The nearest number to 0.00765 in the table is 0.00693, so the corresponding depth of flow is  $4\frac{1}{2}$  in. and the width is 9 in. Adding 50 per cent. to the

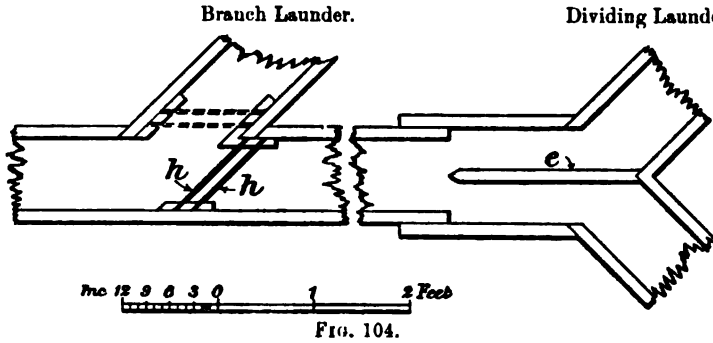


FIG. 104.

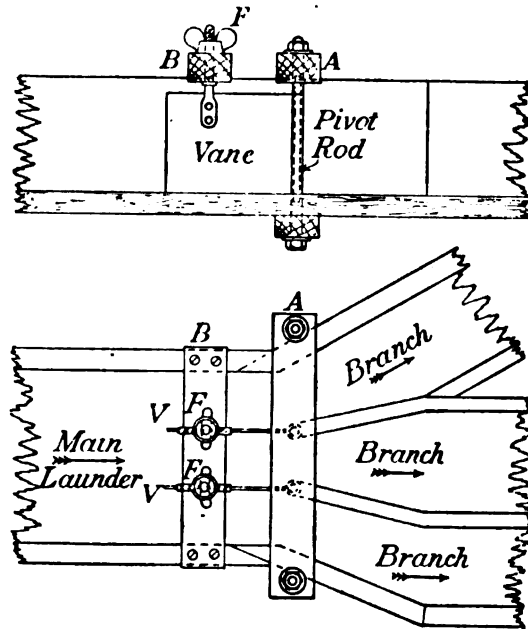


FIG. 105.—Launder with Deflecting Vanes.

depth, the actual dimensions of the launder will be, say,  $7\frac{1}{2}$  inches deep by 9 in. wide, as in Fig. 107. In Table LX. similar figures are given to facilitate the use of equation (7).

For the same quantity of pulp delivered from an air-lift, 9 inches by 9 inches would be a suitable size.

**Branch Launders.**—Single branches may be taken off, as shown in Fig. 104 (left-hand side), at an angle of  $45^\circ$ . When the pulp is to be equally divided

between two branch launders, as in Fig. 104 (right-hand side), a dividing board *e* should be fixed in the main launder so that the division takes place at a point where the flow is steady.

Movable deflecting vanes are sometimes used at the point of division as *V V* in Fig. 105, instead of the fixed divider shown in Fig. 104. By this means the division of pulp between the several branches can be varied as desired.

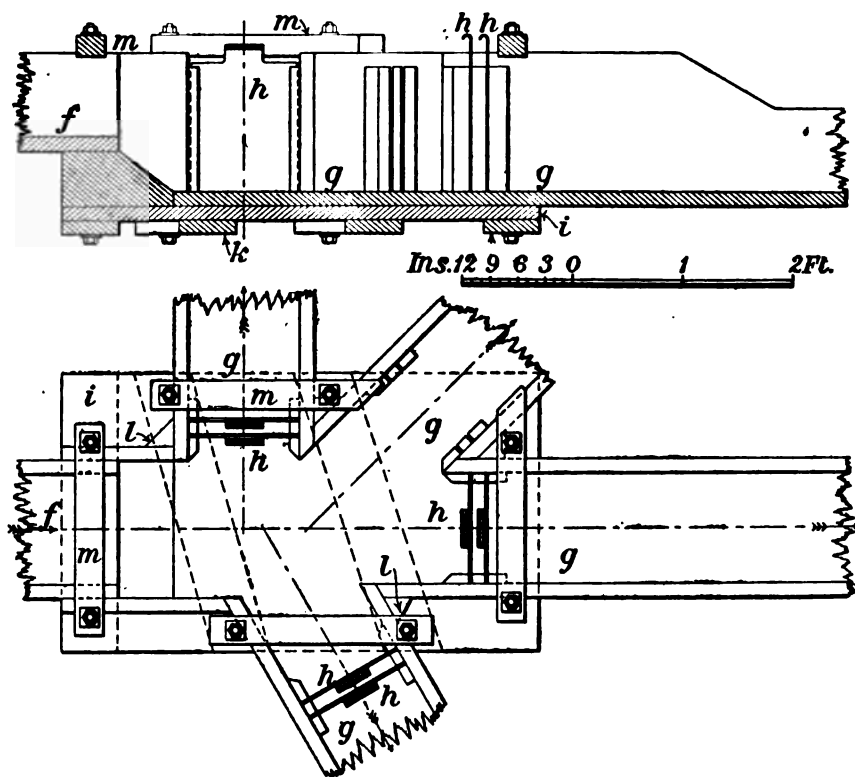


FIG. 106.—Plan of Distributing Box.

Wood strips or iron plates are fixed across the launders at *A* and *B*. The vertical iron pivot rods turn in sockets fixed in *A* and in the launder bottom. The outer end of each vane *V* carries a screwed rod which projects up through a curved slot in *B*. The vane is held in any position by the flynut *F* shown. In this case the bottom of the main launder *f* should be about 6 in. higher than the level of the branches *g g*, this extra head being allowed to scour a clear channel to any branch that may be open for the time being. The sides of the branch launders for the first few feet should be as high as the sides of the main launder, on account of the splashing which takes place. Also in the arrangement illustrated in Fig. 104 (left-hand side), the sides of main and branch launders should be raised about 6 in.

more than their normal height, for a length of 18 in. or 2 ft. below the point of deflection.

Sliding gates *h h* of sheet steel are commonly used for closing the branches, and should always be arranged in pairs as shown. Any sand or slime which leaks around the edges of the first gate settles in the space between the two, and quickly forms a water-tight seal. Consequently the gates may be made an easy fit in the grooves.

The distributing box is constructed in the following manner: A base is built up of  $1\frac{1}{2}$ -in. planks *i* (Fig. 106) spiked to battens *k*. The ends of the

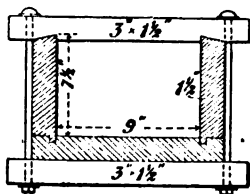


FIG. 107.—Launder.

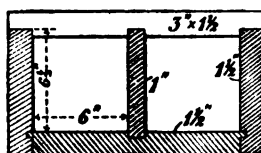


FIG. 108.—Launder.

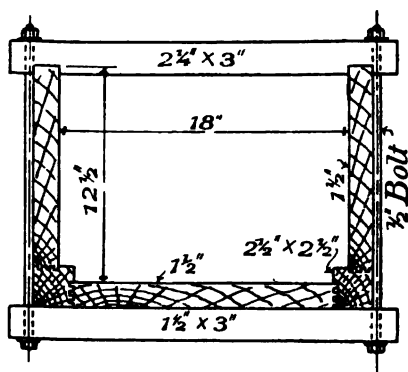


FIG. 109.

branch launders are screwed to this base, and the joints between them made good by triangular fillets *l l*. They are also clamped to the base by bolts, which pass through the timber strips *m* at the top and the battens *k k* below. Sometimes a trough is formed in the bottom of the box just in front of the main launder, so that the inflowing pulp impinges upon a deposit of sand which settles in the trough.

**Construction of Wooden Launders.**—Several variations in use are shown in Figs. 107 to 113 which, being fully dimensioned, require little explanation. The clamps in Figs. 107, 110, and 111 may be spaced from 6 to 10 ft. apart, according to size of launder and thickness of material.

When several launders have to be carried side by side, they may with economy be combined as in Fig. 108. The top distance pieces may be from 12 to 16 ft. apart, and provided with clamps of the type shown in Fig. 109, placed midway between them. Fig. 109 shows a specially elaborate method

of making the joint between side and bottom. End joints between the several lengths are sometimes made by halving the ends of sides and bottoms, and sometimes by butting the ends of sections and making the

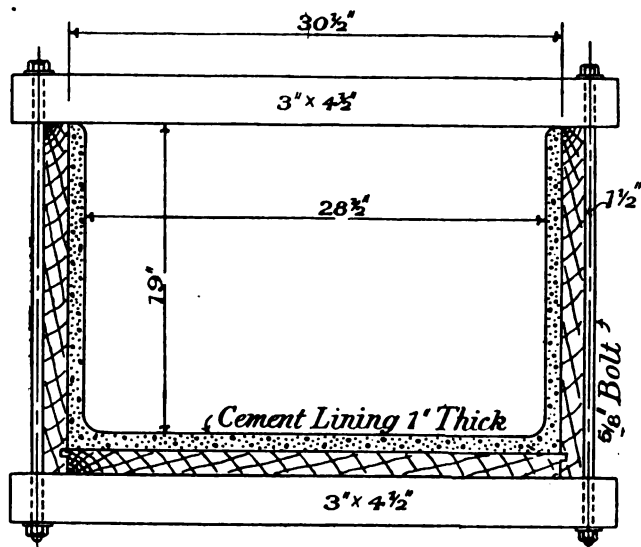


FIG. 110.—Wooden Launder lined with Cement.

joint good with cover strips on the outside. Each method is a satisfactory one. The usual materials for rectangular launders are Baltic deals, Oregon pine, American redwood or clear pine, the latter being preferred on the Witwatersrand, and redwood in many parts of America.

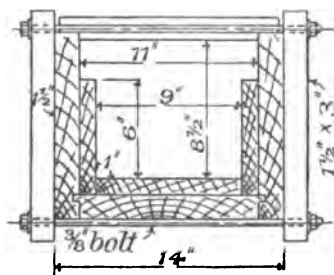


FIG. 111.—Launder with hardwood lining.

The sides of curved launders are made in two ways, viz. by using several thicknesses of  $\frac{1}{2}$ -in. lumber bent to the required radii and screwed together, or by using one  $1\frac{1}{2}$ -in. board with a number of vertical sawcuts about half through the thickness from the inside, to allow for the necessary bending.

**Lining Wooden Launders.** — Launders for pulp containing sand or concentrate should be lined with strips of hardwood or other material suitable for resisting abrasion. Cast iron, rubber belting, or concrete is used for this purpose. For the last-named material, wire-netting is stapled to the inside of the lumber to form a key to hold the concrete in place, or the netting may be stretched on fillets of wood nailed to the sides at about 12-in. intervals, these fillets being half the thickness of the concrete lining. A mixture of 3 of sand and 1 of cement is suitable for the purpose.

Asphalt is also used for lining sand launders. The side linings need not be higher than two-thirds the depth of the launder.

At the Boston Consolidated Copper Company's Mill,<sup>1</sup> plate-glass bottom liners have been used, from  $\frac{1}{4}$  to  $\frac{5}{8}$  in. thick, for 26-mesh crushed ore. The wear is said to depend on the size and number of the largest particles, the limit being stated at 3.5 mm. (0.14 in.). These plates are held down by quarter-round fillets nailed in the corners of the launders. For launders carrying slime no lining is required.

**Concrete Linings for Launders.**—According to F. Wartenweiler,<sup>2</sup> at the Consolidated Main Reef plant at Johannesburg, a concrete liner introduced by J. T. Walker has replaced all others for some years. At first concrete U-cross-section blocks were placed in the mill launders. For a period of three years only the lining on the bends has been replaced. The wider portions of the launder taking pulp from mill pumps to tube-mill cones (slope  $9\frac{1}{2}$  per cent.) were lined, bottoms and sides, with  $2\frac{1}{2}$ -in. thick, flat concrete slabs and the narrower parts with U-shaped blocks.

TABLE LXI.—*Life of Concrete Lining for Launders.*

| Description.   | Slope of Launders, per cent. | Screen Gradings of Pulp. |       |       | Life to Date.          |
|--|------------------------------|--------------------------|-------|-------|------------------------|
|  |                              | +60                      | +90   | −90   |                        |
| Mill launders (U-blocks) . . .   | $7\frac{1}{2}$               | 53.55                    | 11.38 | 35.07 | 3 years +              |
| Wide launders conveying pulp from pump discharge to tube-mill cones (slabs) . . .    | $9\frac{1}{2}$               | 53.55                    | 11.38 | 35.07 | 3 years +              |
| Wide launders conveying pulp from pump discharge to tube-mill cones (U-blocks) . . . | $9\frac{1}{2}$               | ...                      | ...   | ...   | ...                    |
| Overflow launders from tube-mill diaphragm cones (U-blocks) . . .                    | 18                           | 9.6                      | 22.4  | 68.0  | $2\frac{1}{2}$ years   |
| Overflow launders from tube-mill diaphragm cones (bends) . . .                       | 18                           | 9.6                      | 22.4  | 68.0  | 4 months               |
| Launders leaving amalgamation tables (plate house) (U-blocks) . . .                  | 6                            | 14.2                     | 34.4  | 51.4  | 4 years +              |
| Return classifier cones to mill-pulp elevating pumps (U-blocks) . . .                | $8\frac{1}{2}$               | 56.8                     | 26.8  | 16.4  | $2\frac{1}{2}$ years + |
| Launder under return classifier cone discharge (U-blocks) . . .                      | 10                           | 56.8                     | 26.8  | 16.4  | 5 months               |

*Note.*—The + sign has been used to denote a longer life than that recorded, the liner still being in good condition at the time.

It has been necessary to renew few slabs or blocks in the last three years. Plate-house launders (slope 6 per cent.) lining of U-shaped blocks show little wear after four years' service. The U-blocks in the launder conveying the return pulp (slope  $8\frac{1}{2}$  per cent.) have given two and a half years of service,

<sup>1</sup> Claude T. Rice, *Eng. and Min. Jour.*, April 22, 1911.

<sup>2</sup> *Jour. Chem. Met. and Min. Soc. of S. Africa*, Nov. 1913.



with promise of many additional years of life. At the head of this launder, where the classifier underflows impinge on the lining, the U-blocks last only five months. The following mixture is used for this concrete liner:

One part (by volume) dump waste (mostly quartzite) crushed to  $\frac{1}{2}$  in.

One part (by volume) fine from above crushing (passing  $\frac{1}{4}$  mesh).

One part (by volume) washed drift sand.

One part (by volume) Pretoria Portland cement.

The blocks are cast in a long frame holding 10 to 12 moulds. After allowing to set and dry for two days in the frame, they are submerged in water for six or seven days and are then ready for use.

The flat slabs are usually  $2\frac{1}{2}$  in. thick, with length and width to suit requirement. The blocks or section of U-cross-section are usually 18 in. long and 9 in. deep and from  $11\frac{1}{2}$  in. to  $13\frac{1}{2}$  in. wide over-all. The thickness at the bottom is from 2 in. to  $2\frac{1}{4}$  in., tapering to 1-in. sides at the top. Blocks to fit bends in the launder are made with the same cross-section dimensions. The average weight of an 18-in. length section is 75 pounds.

The slabs and U-blocks are jointed in the launders with quick-setting magnesia cement, or with mill blanketing. Either method is satisfactory. When making renewals, the worn blocks are chipped out and replaced by new ones. The discarded blocks are ground in the tube mills and pass into the pulp circuit.

**Steel Launder.**—These are chiefly suitable for a long straight run without branches, and are generally shaped as in Figs. 112 and 113. They

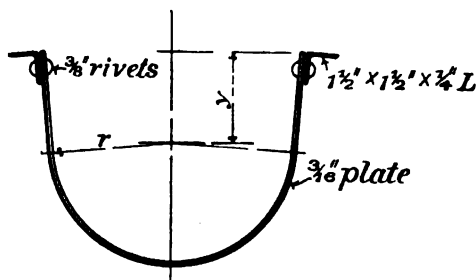


FIG. 112.—Steel Launder.

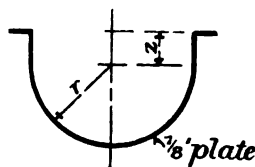


FIG. 113.—Steel Launder.

are used chiefly for slime. The required size can be determined in the following manner: the best theoretical section of flow is a semicircle in which the area equals  $\frac{\pi r^2}{2}$ , the wetted perimeter equals  $\pi r$ , and the mean hydraulic depth equals  $\frac{r}{2}$ .

Consequently:

$$r = \sqrt[5]{\frac{0.81 Q^3}{C^3 S}} \quad (8)$$

Table LIX. can be used with equation (8) in the same way as for equation (6), but the figures in the first column give the radius in inches for equation (8).

To add 50 per cent. excess capacity to the launder, the height of the sides above the semi-circle ( $z$  in Fig. 113) must equal  $0.39 r$ . To add 100 per cent. capacity,  $z$  will be  $0.7854 r$ .

When the launder has spreading sides, inclined at  $12^\circ$  to the vertical, for an extra 50 per cent. make  $y$  equal to  $0.72 r$ .

Rim launders for tanks, cones, and the like, and the distributing troughs in use with zinc boxes are generally made deep and narrow, in order to get as much depth of liquid as possible over the outlet pipes. The general formulæ (4) and (5) must be used for these and other exceptional cases. The flow to be provided in the case of circular rim launders is equal to the total flow divided by twice the number of outlets.

**Grade of Launders.**—The figures in Table LXII. have been obtained by plotting known grades for certain materials and scaling from curves drawn through the plotted points.

TABLE LXII. — *Grades for Launders.*

| Pulp Dilution. | Percentage Moisture. | Size of Predominating Grains. |                    |                  |                   |                  |
|----------------|----------------------|-------------------------------|--------------------|------------------|-------------------|------------------|
|                |                      | Slims.<br>0.0025.             | 100-mesh<br>0.005. | 50-mesh<br>0.01. | 20-mesh<br>0.025. | 10-mesh<br>0.05. |
|                |                      | Percentage Grade of Launder.  |                    |                  |                   |                  |
| ...            | 30                   | ...                           | 30                 | ...              | ...               | ...              |
| ...            | 40                   | ...                           | 25                 | 26               | ...               | ...              |
| 1 to 1         | 50                   | ...                           | 20.3               | 21.3             | 22.6              | ...              |
| 1½ to 1        | 60                   | ...                           | 15.5               | 16.5             | 17.9              | 19.8             |
| 2 to 1         | 67                   | ...                           | 12.5               | 13.5             | 14.9              | 16.8             |
| 3 to 1         | 75                   | ...                           | 8.8                | 9.8              | 11.4              | 13.4             |
| 4 to 1         | 80                   | ...                           | 6.7                | 7.7              | 9.3               | 11.4             |
| 6 to 1         | 86                   | ...                           | 4.1                | 5.1              | 6.9               | 9.0              |
| 8 to 1         | 89                   | ...                           | 2.8                | 3.8              | 5.6               | 7.7              |
| 10 to 1        | 91                   | 1.5                           | 2.0                | 3.0              | 4.8               | 7.0              |

## CHAPTER XXXVII.

### PUMPS, ELEVATORS, AND AIR LIFTS.

**Pumping Systems.**<sup>1</sup>—The selection of a suitable pumping system involves consideration of several important factors. Owing to the varied conditions of the work and the different characteristics of the fluids to be elevated, the principal requirement is suitability: the system decided upon should involve minimum attention for control and repair, and should operate at a reasonable efficiency. Elevation of pulp and solution are phases of the work in which continuity of operation is of primary importance, and for this reason every opportunity should be taken in the first instance to install a machine that will do the work without unnecessary attention, and without excessive loss of time for repairs and renewals.

It is convenient to subdivide the apparatus more commonly used for this purpose into nine divisions, according to the class of material to be handled, and the height to which it must be elevated. Table LXIIA. shows the applicability of eight classes of pumps for different kinds of work. A selection is usually made from among these.

For low-lift water or solution pumping it is usual to employ either the centrifugal, the air lift, or the Frenier pump. In point of efficiency with good design they may be considered in the order named. The operation of the air lift or of the Frenier pump ensures an aeration of the water or solution. This is sometimes an advantage, sometimes a disadvantage. In the latter event the centrifugal will be chosen as the most suitable. For compactness, and ability to drive by direct motor connection, the centrifugal will also be favoured. For fairly efficient operation with a variable supply and with automatic control the air lift may be considered as superior to either of the other machines mentioned. Where compressed air is not available, the choice will lie between the centrifugal and the Frenier, with a probable decision in favour of the former, on account of greater compactness, higher efficiency, and larger capacity. For small-capacity, low-lift circulating work a single-plunger pump, or diaphragm pump, is sometimes used, and arranged for variable stroke by means of an eccentric adjustment.

For medium-lift pumping there is little to choose between the centrifugal and the compound air lift, although the former is generally used on account of greater compactness. For high-lift work the compound centrifugal or the plunger pump is suitable. The greater comparative flexibility of the latter under the varying conditions of speed and lift indicates a point of superiority over the centrifugal. Multiple-throw plunger pumps are to be preferred,

<sup>1</sup> A. W. Allen, *Met. and Chem. Eng.*, Dec. 1, 1917.

otherwise a heavy flywheel is advisable. A submerged suction saves time, especially in the case where an incrustation is liable to form on the valves, which results in the pump frequently losing its suction.

TABLE LXIIA.

| Material to be Pumped. | Low Lift<br>(to 25 feet).                    | Medium Lift<br>(to 50 feet).          | High Lift<br>(to 100 feet or more).                            |
|------------------------|--|---------------------------------------|--|
| Water or solution      | air lift<br>Frenier pump<br>centrifugal pump | centrifugal pump<br>compound air lift | plunger pump (1-, 2-, or 3-throw)<br>compound centrifugal pump |
| Slime pulp . . .       | air lift<br>diaphragm pump<br>Frenier pump   | compound air lift<br>centrifugal pump | plunger pump (1-, 2-, or 3-throw)                              |
| Sand pulp . . .        | centrifugal pump<br>air lift<br>Frenier pump | centrifugal pump<br>tailing wheel     | plunger pump (3-throw)   |

**Pumping Sand and Slime.**—For this purpose the ordinary centrifugal is largely used. For sand, as well as for slime containing sand, the pumps are provided with a renewable liner, generally made of manganese steel, and the impeller is made of the same material. When used for thick pulp the efficiency is much less than with solution or water, probably on account of the increased internal resistance, due to the viscosity of the pulp. This is shown forcibly by the following figures relating to the power required :

On the basis of 1 ton lifted per hour, the horse-power required for raising water or solution may vary from 0.0015 to 0.0017 for each foot of lift ; whereas for raising 3 to 1 slime pulp, careful tests give 0.002. For 2 to 1 pulp the figure is 0.003 ; and the power increases rapidly, with the proportion of solid matter present, to 0.005 for 1 to 1 pulp. These figures apply strictly to pumps of 10 to 12 in. diameter. Smaller pumps require still more power for the same effective work.

**Special Features of Centrifugals.**—With reciprocating pumps there is a fairly constant efficiency through a large range of capacity, but with centrifugals each type of pulp works at the greatest efficiency only with certain definite relations between quantity, head, and speed of rotation. As a rule, the first two items—the quantity of liquid to be handled and the height to which it is to be lifted—are fixed by the conditions of service. Then, if the maximum efficiency is desired, the size of the pump and the speed must be determined according to the idiosyncrasies of the particular type selected. If a matter of 5 per cent. difference in efficiency may be disregarded, the size of the pump may also be determined beforehand.

**Speed of Centrifugals.**—Centrifugal pumps are particularly sensitive to change of speed. For instance, with an 8-in. Sulzer pump an increase of only 5 per cent. in the speed, from 1000 to 1050 r.p.m., gave, against the same head of 52 ft., an increase of 43 per cent. in the quantity delivered. An increase of 10 per cent. in the speed, from 1500 to 1650 r.p.m., against a head of 115 ft. increased the delivery by 58 per cent.

These data show the necessity for driving a centrifugal or turbine pump exactly at the right speed for the work to be done. This is especially true in the case of circulating pumps, where the quantity delivered cannot be readily measured. As a rule, it is better to specify the conditions of service, and let the maker be responsible for the speed required.

It follows, therefore, that no absolute rules can be given for the speed of centrifugal pumps; but, as it is often convenient in preliminary arrangements of plant, to have an approximation thereto, the following table is given, representing common practice for high-speed and low-speed pumps for various sizes and heads, the latter type being preferred in cyanide plants:

TABLE LXIII. — *Revolutions per Minute for Volute Centrifugal Pumps.*

| Diam. of<br>Outlet.<br>Inches. | 20 feet Head |             | 40 feet Head. |             | 60 feet Head. |             |
|--------------------------------|--------------|-------------|---------------|-------------|---------------|-------------|
|                                | Slow Speed.  | High Speed. | Slow Speed.   | High Speed. | Slow Speed.   | High Speed. |
| 2                              | 715          | 1380        | 1005          | 1780        | 1230          | 2100        |
| 3                              | 650          | 1270        | 910           | 1610        | 1110          | 1890        |
| 4                              | 585          | 1160        | 820           | 1455        | 1000          | 1700        |
| 5                              | 530          | 1070        | 740           | 1325        | 905           | 1540        |
| 6                              | 475          | 990         | 670           | 1210        | 820           | 1400        |
| 7                              | 430          | 925         | 610           | 1115        | 750           | 1280        |
| 8                              | 380          | 870         | 555           | 1040        | 685           | 1190        |
| 10                             | 340          | 790         | 480           | 940         | 590           | 1080        |
| 12                             | 315          | 750         | 440           | 890         | 535           | 1020        |

**Capacity of Centrifugal Pumps.** — Centrifugal pumps are generally catalogued with nominal capacity according to the diameter of the outlet. The capacity can be increased by an increase of the speed, but it is better practice to use a pump designed to give its maximum efficiency with a moderate speed and delivery, because this minimises wear and tear, and consequent liability to frequent repairs.

Table LXIV. is based on the average results from several well-known makes.

TABLE LXIV. — *Capacity of Centrifugal Pumps for Water or Solution.*

| Diameter.<br>Inches. | Cubic Feet per hour. |        | Short Tons (2000 lb.)<br>per hour. |     |
|----------------------|----------------------|--------|------------------------------------|-----|
|                      | From                 | To     | From                               | To  |
| 1½                   | 240                  | 380    | 7.5                                | 12  |
| 2                    | 480                  | 670    | 15                                 | 20  |
| 3                    | 960                  | 1,540  | 30                                 | 50  |
| 4                    | 1,920                | 2,640  | 60                                 | 80  |
| 5                    | 3,180                | 4,130  | 100                                | 130 |
| 6                    | 4,810                | 5,860  | 150                                | 180 |
| 7                    | 6,450                | 7,970  | 200                                | 250 |
| 8                    | 8,660                | 10,370 | 270                                | 320 |
| 10                   | 13,730               | 16,130 | 430                                | 500 |
| 12                   | 20,060               | 23,040 | 630                                | 720 |

**Ram Pumps.** — For vacuum leaching it is common to use three-throw ram pumps, both for air and solution. They are also largely employed in the

vacuum methods of filtration, and for filling and washing filter presses and pressure filters. Single and 2-in. plunger pumps are also used for the latter purpose. The capacity of this type is proportional to the speed, and can be calculated by multiplying the area of the plungers by the length of stroke and by the number of strokes per minute. The speed of the plunger is usually between 80 ft. per minute for a 5-in. stroke, to 120 ft. per minute for a 10-in. stroke.

At many mines, especially in Australia, double-throw plunger pumps are used for lifting battery pulp, with satisfactory results. Figs. 114 to 116 show a simply designed tailing pump, such as may be made in small engineering works usually found in mining districts. The dimensions given are for cast iron; the spur gear, connecting rods and crank shaft are of steel, and the timber frame is of hard wood. The lift may be anything up to 150 ft.

The chief difference between tailing pumps and water pumps is that in the former all the passages have to be choked, so as to give a high velocity to the flow of pulp through the passages and pipes, without unduly increasing the speed of the pump, the obvious object being to prevent the sand settling while it is being transferred. Also, it will be noticed in Figs. 114 and 116 that there is a half-round groove *a* just below the stuffing-box, which is connected with a pipe *b*, provided with a valve *c*, and below this groove is a cup leather which fits tightly around the plunger. The pipe *b* is connected with a cistern of clean water fixed at a suitable height, so that the water may flow by gravity. In the up-stroke of the plunger the valve *c* opens and admits a supply of clean water to the groove *a*, which then runs down between the plunger and the cup leather, washing off any adhering grit, which is thus prevented from reaching the packing. On the down-stroke, the valve *c* closes by the pressure from inside, and the cup leather is pressed tightly around the plunger, and thus prevents grit being forced into the groove or packing. The valve box shown is arranged for ordinary clack valves, but some makers prefer to use disc valves, for which a box can be easily designed. An air vessel should be provided to keep the flow more regular, and to relieve sudden internal shocks to which these pumps are liable. The delivery pipe is attached to *d*, and must be in a regular line, free from parts where sand can lodge. A sluice valve *e* should be provided to empty all the passages immediately the pump stops. A fairly high speed of 15 to 25 strokes a minute is desirable, and the capacity may be regulated, in case of need, by adjusting the crank pin, which is preferable to varying the speed, as a short, quick stroke prevents settling better than a long, slow one. Leather may be used for the valves, but rubber will be found more durable. Old rubber belting answers the purpose well, and lasts three to four months. Plunger cup leathers last about a year.

It is not desirable to make the suction pipe more than 8 ft. long. The sumps should be small, but large enough for a man to get into and clear. A fall of 2 or 3 ft. for the pulp into the sumps prevents settlement.

The following examples illustrate the sizes used in practice, from which others can be readily calculated, for any given case. At the Bonanza mine,

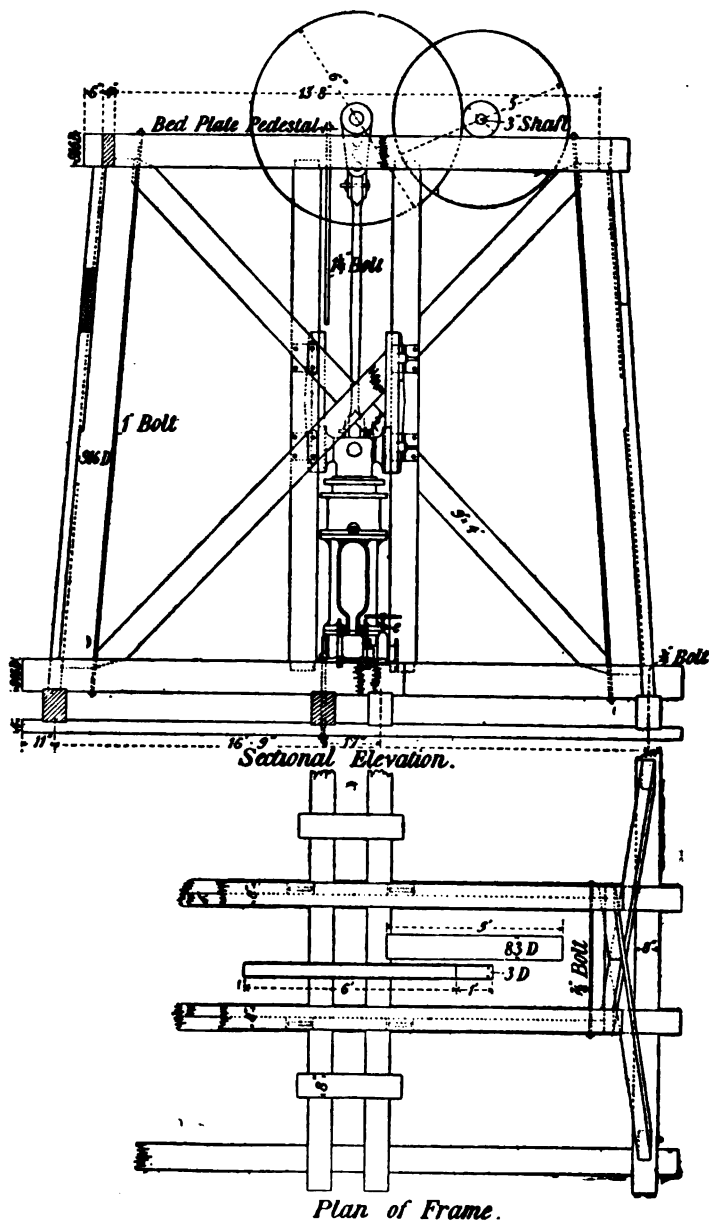


FIG. 114.—Tailing Pump; Side View of Frame, etc.

Johannesburg, for 55 stamps, crushing about 260 tons per day, one double-throw plunger pump, 12-in. diameter by 48-in. stroke, lifts the whole pulp,

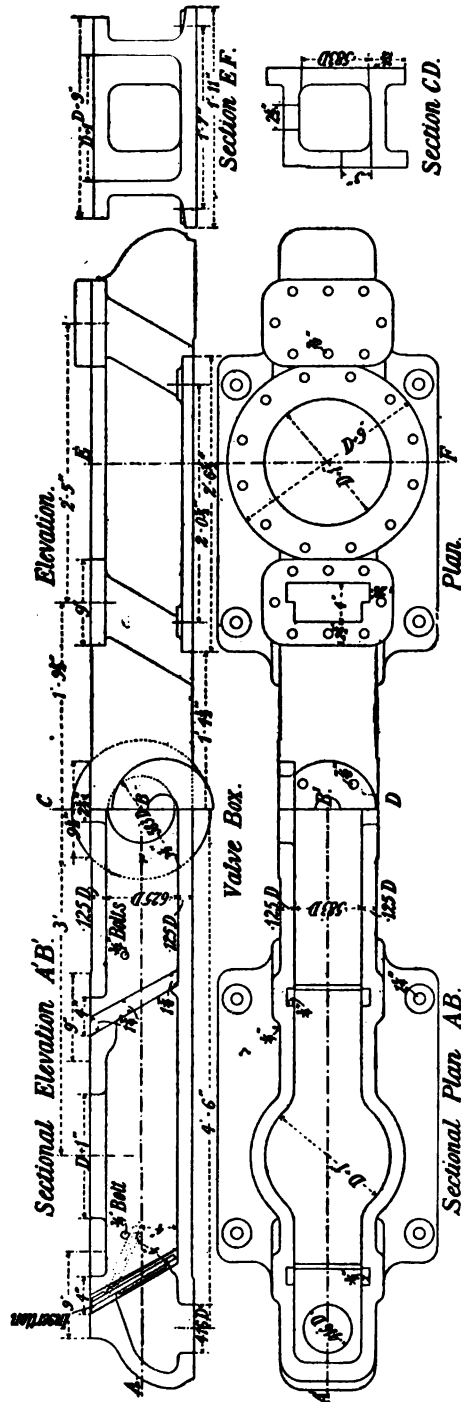


FIG. 115.—Valve Box for Tailing Pump in detail; D equals Diameter of Plunger.





say 2600 tons, to a height of 45 ft. A second pump is provided as a standby, and to allow of repairs being effected without interfering with the work.

At the May Consolidated mine, Johannesburg, one double-throw plunger pump, 15 in. diameter, lifts the pulp from 100 stamps to a height of 65 ft. These two examples give a mean figure of 1 circular inch of plunger area for each 20 tons of pulp to be lifted per day.

**Frenier Spiral Sand Pump.**—This is largely used, as a standby, for elevating tailings or slimes.

Fig. 117 shows a vertical section through the main part, called the 'pump wheel,' which consists of a spiral tube made of a coiled strip of steel, 6, 8, or 10 in. broad, secured between two steel discs, the joints being made air tight.

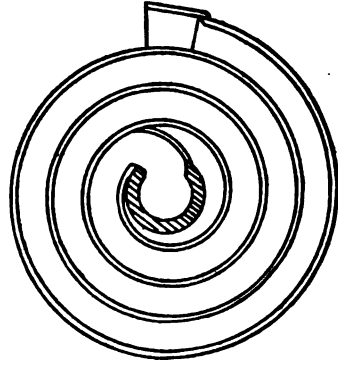


FIG. 117.—Vertical Section through Pump Wheel.

The pump wheel A, Fig. 118, has a shaft B at its centre, one end being hollow, which connects the centre of the spiral tube with the delivery pipe C, through a detachable tube D, provided with a stuffing-box. On the other end of the shaft spur gearing is keyed, to drive the pump. E is the sump box into which the pulp

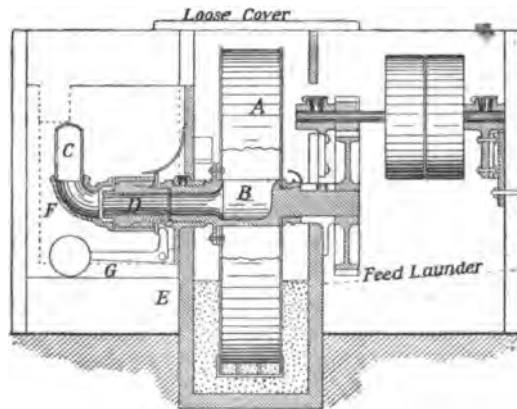


FIG. 118.—Frenier Sand Pump; End Sectional Elevation.

runs, and has an overflow 7 in. below the shaft. As the wheel turns, its mouth scoops up some pulp into the outer portion of the spiral tube, and as rotation continues this flows on towards the centre. Between the points where the mouth leaves the pulp and enters it again the tube takes up air, which becomes locked in by the next scoop of pulp, thus producing layers, as it were, of air and pulp flowing towards the delivery pipe. The elevating action depends upon the hydrostatic heads in each turn of the

spiral, and this is assisted by the air in its attempt to ascend through the column of pulp in the delivery pipe, as in the case of air lifts. The pressure produced in the scooping up of the pulp also assists. The pump works most efficiently when the speed is slow, about 20 r.p.m. There are no valves, and the part subject to greatest wear is between the rotating tube D and the bend F, which may be replaced at a low cost. A weighted lever G is used to balance the thrust produced by the delivery pipe.

The wheels are made 44, 48, and 54 in. in diameter, which lift 14, 18, and 24 ft., respectively, for altitudes up to 1000 ft., and 3 in. less for every thousand feet higher.

The operation of a Frenier pump is essentially that of an air-lift, a certain amount of air being compressed at each revolution of the spiral. No increase in the capacity of the pump can be attained by an increase in the speed of the spiral, and no departure should be made from the determined ratio. There is no possibility of obtaining excess air pressure, and it is therefore imperative that the discharge should be free and unhindered. To this end the precaution observed in the lay-out of an air-lift installation should be rigidly followed, and all bends and elbows avoided in the lift pipe. It is necessary that the air be allowed to escape as soon as the lifting is accomplished, and the best way to ensure this is to provide a vertical discharge directly into the bottom of a short launder having a sharp fall. In operating a pump of this kind it is necessary to keep the pulp in the box at a definite level. Where it is impracticable to arrange for the pump to have its exact load, maximum efficiency and minimum of trouble will result if an overflow exit is provided at the fixed pulp level. The excess pulp can then be handled by a separate arrangement such as a variable-feed self-operating air lift, which will require power in almost direct proportion to the amount of excess pulp handled.

**The Air Lift.**—This apparatus, which is now much used for lifting pulp, consists essentially of three parts, viz. (1) a well into which the pulp is fed, (2) a rising main from the bottom of the well to the point of delivery, and (3) a pipe to supply air to the lower end of the rising main.

There is a considerable variation in the proportions in different districts. For instance, in Western Australia the depth of the well is usually equal to the useful lift; in South Africa it is generally  $1\frac{1}{2}$  times, and in many other places it is twice the useful lift.

**Quantity of Air required.**—The motive force of the lift is the difference between the pressure of the short column of pulp in the well, and the pressure of the longer column of mixed air and pulp in the rising main. For the present it may be assumed that these are equal. Then let

H = height, in feet, of rising main.

h = depth, in feet, of column in well, at any moment.

x = length, in feet, of a portion of the rising main, equal to the volume of air in the whole of it.

G = specific gravity of pulp.

g = specific gravity of air.

The pressure due to the column in the well is its weight per square inch of sectional area, that is

$$0.433Gh,$$

and the pressure of air and pulp in the rising main is similarly

$$0.433(H-x)G + 0.433xg.$$

Equating these two pressures and transposing, then

$$G(H-h) = x(G-g) \quad (4)$$

The specific gravity of air at ordinary pressure is 0.00123, and even at 45 lb. gauge pressure it is only 0.005, whereas that of the pulp is always greater than 1; therefore the omission of  $g$  from the right-hand side of equation (4) will make an error of only 0.1 to 0.5 per cent., which may for practical purposes be neglected. This is equivalent to assuming that the air has no weight, and the equation becomes

$$x = H - h \quad (5)$$

That is to say, the depth of pulp in each column is exactly the same and is equal to  $h$ , while the extra length in the long column is occupied by air. Consequently the ratio by volume between air and pulp is

$$\frac{H-h}{h},$$

and if  $v$  = cubic feet of air required, theoretically, to balance 1 ton of pulp, then

$$v = \frac{32(H-h)}{Gh} \quad (6)$$

Hence for a given useful lift, the deeper the well, the smaller will be the volume of air in the rising main; but as the pressure is greater, more free air is required for equal volumes of compressed air, and relatively more power will be consumed in compression. Where a separate compressor is used to give the pressure which corresponds to the depth of well, a deep well requires more power than a shallow one for a given net lift. Thus for a 20-ft. lift a 60-ft. well would require about 22 per cent. more power than a 20-ft. well, but this consideration is of little importance as compared with steadiness in working.

On the other hand, when the air is taken from a high-pressure source, as is often the case, a deep well is more economical than a shallow one, because less air is used per ton of pulp lifted.

For the same reason it follows that with any given depth of well, the apparatus works with the greatest efficiency when the well is full.

Let  $V$  = cubic feet of free air to be compressed per ton of pulp.

$P$  = atmospheric pressure, say 14.7 lb. per sq. in.

$p$  = mean absolute pressure in rising main =  $14.7 + \frac{0.433Gh}{14.7}$  lb.

Then, because

$$V = \frac{vp}{P}$$

by substituting the values given above

$$V = \frac{H-h}{Gh} (32 + 0.471Gh). \quad (7)$$

This quantity of free air is quite independent of the pressure of the compressing plant, and the lift can only fairly be debited with the power required to compress this volume of air to the volume  $= \frac{v}{2}$ , and pressure  $2p$ , at which it is delivered to the bottom of the well.

These equations refer, of course, only to the theoretical volumes of air.

One of the chief losses of efficiency seems to be due to the use of much higher pressures than are necessary, and to the air, in some cases, bubbling up through the pulp faster than the latter is travelling.

**Velocity of Flow in Rising Main.**—This, in practice, varies from 0.75 ft. to 6 ft. per second in the various cases that have been examined. It seems likely that with the higher velocity there is less loss by bubbles of air rising faster than the pulp. From some rough experiments it appears that air bubbles about  $\frac{1}{4}$ -in. in diameter rise at the rate of about 0.75 ft. per second in water. This speed, therefore, would have a much greater relative importance at the lowest velocity mentioned above than it would at the highest rate of flow.

**Constructional Details.**—In South Africa the rising main is sometimes lined inside with 1-in. timber lagging. A strong built-up timber block is fixed a little above the top of the pipe, to receive the impact of the issuing pulp, and to deflect it downwards into the receiving box.

The wells are usually circular, lined with concrete and finished inside with cement rendering. The diameter is large enough to allow a man to descend easily when both pipes are in position, say 5 ft. for a 12-in. main.

In Western Australia and the United States the well usually consists of an iron cylinder coned at the bottom, or in some cases the well is simply a hole bored into the rock formation.

When the inlet end gets choked, it is often cleared by plugging the discharge end of the rising main for a few minutes; the back pressure of the air then reverses the flow through the well and removes the obstruction.

**Size of Pipes.**—The diameter of the rising main can be calculated by the ordinary rules, according to the volume of pulp and air passing through it, and the rate of flow desired. The size of the air pipe depends not only upon the quantity of air delivered, but also upon the distance from the source of supply. Where an air receiver is placed close to the lift, a 4-in. pipe for a 10-in. lift gives good results.

In Western Australia an 8-in. lift raising pulp 37 ft. with a well also 37 ft., worked better with a 2-in. than with a 1-in. air pipe.<sup>1</sup> But at the working pressure stated, namely, 8 lb. per sq. inch, it is evident that the lift was working with the well only half full.

<sup>1</sup> Edwin O. Watt, *Trans. Aust. Inst. Min. Eng.*, vol. ix, part i. p. 9.

**Efficiency of Air Lifts.**—C. T. A. Hansen<sup>1</sup> says that the air lift is capable of giving a high efficiency. He attributes the chief losses in efficiency, apart from the losses in the compressor, to—

- (a) Excessive friction in the rising main.
- (b) Defective mixing of the air and water.
- (c) Incomplete expansion of the air.

(d) Slow velocity of the mixture in the rising main, the minimum mean rate of flow given being 6 ft. per second.

Referring to the mixture of air and water in the rising main, he says: "Prof. Josse of Berlin has proved experimentally that with a well-designed nozzle the efficiency was 25 per cent. higher than when the air was admitted by a plain cylindrical pipe." He also adds that "for a maximum efficiency it is further essential that the compressed air should be supplied with the lowest possible pressure at the base of the rising pipe, and escape at the top as nearly as possible with the same pressure as the atmosphere." A bend at the top of the rising pipe interferes with the latter condition by preventing the air, after it has done its work, from escaping freely.

Philip Bjorling<sup>2</sup> says that Prof. Randell, while testing Pohle's original air lift, found the following efficiency:

| When $\frac{H}{h}$<br>equals | Efficiency<br>per cent. |
|------------------------------|-------------------------|
| 0.5                          | 50                      |
| 1                            | 40                      |
| 1.5                          | 30                      |
| 2                            | 25                      |

$h$  equals height in feet of water in well, and  $H$  equals height to which the water is delivered.

**Tailing Wheels.**—These are commonly used on the Witwatersrand gold-field for lifting battery pulp to spitzlutte placed above the collecting vats, more particularly in double-tier plants.

Two kinds of wheel have been used: (a) with outside buckets like an ordinary water wheel, and (b) with internal buckets. The disadvantage of the external-bucket wheel is that it can only pick up the pulp by dipping into it, and can only discharge by throwing the pulp forward into the upper launder, so that the drip and splash are much greater than with a well-designed wheel with internal buckets, into which the pulp can be delivered by a launder without any splashing whatever. Consequently the latter type only is used at the present time.

The mechanical stresses to which a lifting wheel is subjected are similar in nature and degree to those which occur in a water wheel, so that the rules as

<sup>1</sup> *Engineering*, vol. lxxvi. pp. 809-10, 1903.

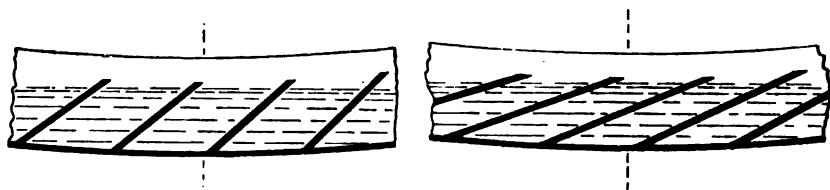
<sup>2</sup> *The Mech. Eng.*, xiii. pp. 341-2, 1904.

to the strength of the various parts laid down in special treatises on the latter may be followed. The chief essential difference is, that in the internal lifting wheel the buckets must laterally overhang their supporting arms in order to allow of the reception and discharge of the pulp.

Double wheels have been built with a set of buckets on each side, and the driving pulley between them, so that they are perfectly balanced laterally.

In the most recent practice the driving pulley is always of large diameter, consequently the stresses developed in transmitting the power from the pulley to the buckets are confined to a small portion of the framing, namely, the part near the outer circumference, the chief function of the radial arms being to transfer the weight of the rim of the wheel to the shaft. A belt or rope drive is generally used.

In deciding upon the diameter necessary for a given lift, allowance must be made for the height necessarily lost between the lip of the bucket in its highest position and the bottom of the receiving launder. Also, for the radial



Figs. 119, 120.—Positions of Buckets.

depth of the buckets, and for the small space between them and the supply launder.

In practice the ratio of outside diameter to net lift is generally about 1.3 to 1, so that about 23 per cent. of the power expended is unavoidably wasted from this cause alone.

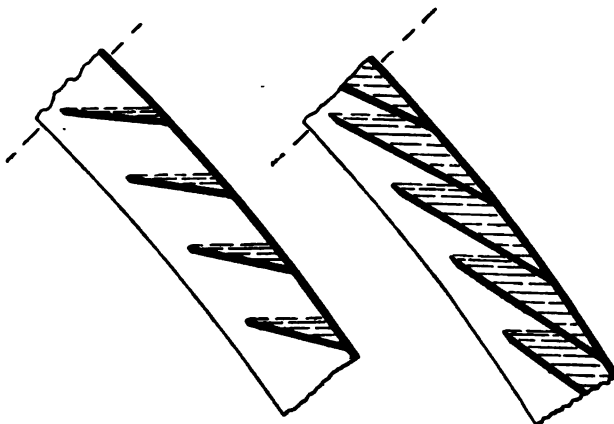
The buckets are generally formed by plane divisions, and some diversity of opinion has been displayed in existing examples as to the proper angle at which these dividing planes should be placed. It is therefore worth while to examine briefly the conditions which affect this element of the design.

For this purpose the operation of the buckets may be considered in three separate stages, namely, filling, lifting, and discharging.

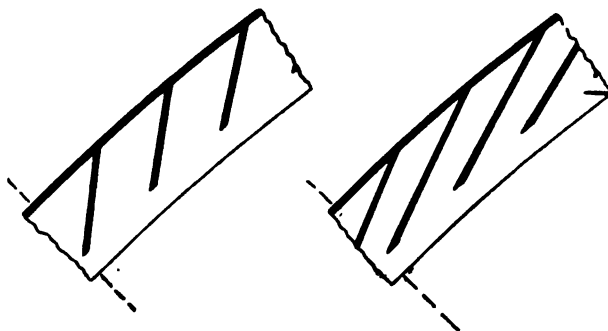
Figs. 119 and 120 show a few buckets in their lowest position for two different cases, that is to say, in Fig. 119 the divisions make a small angle with the radius, and in Fig. 120, a large one. It is at once apparent that the mouth of the bucket in the second case is much narrower than that in the first figure. Consequently, if the angle is made very large, the mouth becomes so extremely narrow that the air inside is liable to be trapped by the entering pulp, so that the bucket could not readily be filled, and the amount of splash would be increased. A wheel designed with a very large angle could, therefore, only be run slowly in order to give the buckets time to fill properly.

In Figs. 121 and 122 the same buckets are shown at the point where the

discharge is intended to begin. In the former figure nearly the whole of the pulp in each bucket has been lost, having dribbled back down the inside of the wheel. To overcome this difficulty, fixed aprons are placed close to the inner edges of the buckets, so that the water cannot so easily escape. In Fig. 122, however, a much larger proportion of the pulp has been lifted to the discharge, so from this point of view the larger angle is far more satisfactory, because there is less drip, and also less power wasted in lifting a portion of the pulp, through part of the height, over and over again.



Figs. 121, 122.—Positions of Buckets.



Figs. 123, 124.—Positions of Buckets.

These drawings, of course, are only correct for wheels moving very slowly, because a greater speed will cause a greater portion of the liquid to be lifted, owing to its inertia. But in any case the difference between the quantities lifted by the two moving forms of buckets will be considerable, and it is better to have a slowly moving wheel working efficiently, than a quickly moving one working badly.

Figs. 123 and 124 show the position of the buckets at the end of the discharge. Here the first named has an apparent advantage, because the dividing planes attain a more vertical position. But the discharge is so rapid in either case that this is of no real benefit except in very high wheels, in



which the sand tends to settle in the bucket before reaching the point of discharge.

By placing the divisions close together, as in Fig. 125, the total quantity lifted can be increased, but only at the risk of some of the sand remaining in the buckets, owing to their mouths being too narrow.

It appears, then, that no absolute rule can be laid down as to the best possible inclination of the bucket divisions, but an angle of  $65^\circ$  may be recommended as avoiding the disadvantages of either extreme. With this angle, a suitable pitch for the buckets may be estimated in the following manner, as shown in Fig. 126, where  $ab$  is the outer circle of the wheel,  $cd$  the inside of the shrouding, and the dotted line  $ef$  represents the inner

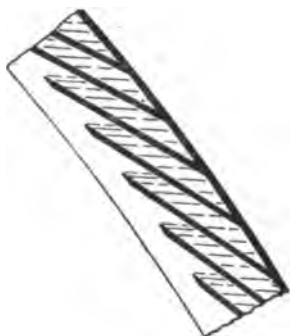


FIG. 125.—Position of Buckets.

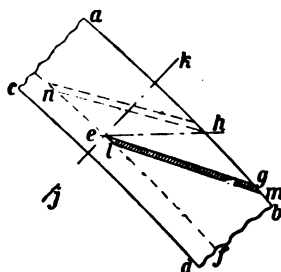


FIG. 126.—Pitch of Buckets.

circumference of the buckets: Let  $jk$  represent a radius of the wheel at the point where discharge commences, and which cuts the circle  $ef$  at the point  $e$ . From  $e$  draw a horizontal line cutting  $ab$  at  $h$ , and draw  $eg$  making an angle of  $65^\circ$  with  $jk$ . This latter line is the inside of one bucket. The thickness of the division and of the hard-wood lining (if any) is marked off by the line  $lm$ , and the distance  $hm$  is the required pitch. As this may not be an exact sub-multiple of the circumference, it must be modified accordingly. It will be seen that the area of the triangle  $ehg$  is nearly equal to half that of the figure  $enhg$ , hence for wheels laid out in this way the capacity per revolution is given with sufficient accuracy by the following formula:

- Let  $R$  = radius in feet at back of buckets.  
 $r$  = radius in feet at mouth of same.  
 $n$  = number of buckets.  
 $l$  = length of divisions measured on incline in inches.  
 $t$  = thickness of same in inches.  
 $B$  = inside width of bucket.  
 $C$  = capacity in cubic feet per revolution.

Then

$$C = B \left( \frac{\pi R^2 - \pi r^2}{2} - \frac{lt n}{144} \right) \quad (2)$$

The quantities  $\pi R^2$  and  $\pi r^2$ , being simply the areas of the respective circles, can be at once written down from tables. When the divisions are thin, as

for instance, those of sheet iron without wooden lining, the term  $\frac{lt\eta}{144}$  may be neglected, and

$$C = B \left( \frac{\pi R^2 - \pi r^2}{2} \right) \quad . \quad . \quad . \quad . \quad . \quad (3)$$

The following rule, by J. Frier, used in America, is given as an example of a very small angle for the buckets: The centre of the receiving launder is at a height above the bottom of the wheel equal to three-quarters of the diameter of the latter. A circle is drawn tangential to the launder, and the buckets are also tangents to this circle. In other words, the diameter of the tangent circle is half that of the wheel; and, consequently, the angle between bucket and radius at the outer end of the bucket is only  $30^\circ$ . This is said to give good results at a peripheral speed of 500 feet per minute.

Table LXV., taken from a paper by W. H. Wood and E. J. Laschinger,<sup>1</sup> may be useful in designing a wheel for a given number of stamps. One stamp is assumed to crush 5.5 tons per 24 hours.

TABLE LXV.

| Ratio by Weight.<br>Water to Rock. | Mill Pulp per Stamp. |                 |
|------------------------------------|----------------------|-----------------|
|                                    | Cub. ft. per min.    | Pounds per min. |
| 6-1                                | 0.7796               | 53.472          |
| 7-1                                | 0.9018               | 61.111          |
| 8-1                                | 1.0241               | 68.750          |
| 9-1                                | 1.1463               | 76.389          |
| 10-1                               | 1.2685               | 84.028          |

In the same paper is given the speed of tailing wheels at which the buckets just cease to discharge their contents due to centrifugal force. This is called the critical or limiting speed; the actual speed in practice being about one-third. Table LXVI. gives the critical speeds and actual speeds in practice.

TABLE LXVI.

| Diameter of<br>Wheel<br>in feet. | Critical Speed. |                                 | Speed in Practice. |                                 |
|----------------------------------|-----------------|---------------------------------|--------------------|---------------------------------|
|                                  | Revs. per min.  | Peripheral<br>vel. ft. per min. | Revs. per min.     | Peripheral<br>vel. ft. per min. |
| 10                               | 24.218          | 761                             | 8.07               | 254                             |
| 20                               | 17.125          | 1076                            | 5.71               | 359                             |
| 30                               | 13.983          | 1318                            | 4.66               | 439                             |
| 40                               | 12.110          | 1522                            | 4.04               | 507                             |
| 50                               | 10.830          | 1701                            | 3.61               | 567                             |
| 60                               | 9.887           | 1863                            | 3.29               | 621                             |
| 70                               | 9.154           | 2013                            | 3.05               | 671                             |

<sup>1</sup> *Eng. and Min. Jour.*, vol. lxxvii. pp. 482-8, 1904.

The time allowed for the buckets to discharge their contents is given as not less than 3 seconds for wheels of about 40 ft. diameter, and 5 seconds for 60-ft. wheels.

**Efficiency of Tailing Wheels.**—Wood and Laschinger<sup>1</sup> made a determination of the mechanical efficiency and give the following data :

Diameter of wheel, 25 ft.  
Lift, 19 ft. 1 in.  
Weight of pulp lifted, 5549 lb. per min.  
Theoretical h.p. required, 3.208.  
Actual h.p. delivered by motor, 6.935.  
Total power efficiency, 48.51 per cent.

This determination involves three speed reductions from the motor, and it is thought that with wheels of greater diameter the efficiency would be higher.

**Construction of Tailing Wheels.**—A few years ago, N. Wilson, consulting mechanical engineer to the East Rand Proprietary Mines, suggested the use of tailing wheels of the bicycle type with tension spokes, which are now often adopted on the Rand. The first wheel of this kind was designed by Messrs Fraser & Chalmers, and erected at the Consolidated Main Reef mine. It is illustrated in Figs. 127 to 130.

The steel axle is 10 ft. long over all by 7 in. in diam. at the centre. Near each end it is turned down to  $6\frac{1}{2}$  in., thus forming shoulders to take the pressure of the two bosses of cast steel, to which the inner ends of the inclined spokes are attached. Each boss has a circular flange inclined at such an angle that the spokes pass squarely through it. They are secured by nuts on the inside of this flange. There are 22 spokes,  $\frac{3}{4}$ -in. diam. with 1-in. ends, on each face of the wheel, those on one face alternating as regards angular position with those on the other face, so that the rim is held at 44 points in its circumference. The outer ends of the spokes pass through two rings of  $3 \times 3 \times \frac{3}{8}$ -in. angle steel, and have flat heads so shaped that they bear truly on the angles. These two rings are 6 in. apart, and the space between them is occupied by hard-wood segments, the whole being drawn up and held together by  $\frac{5}{8}$ -in. bolts at 3-in. centres, which pass through both angles. The construction so far described constitutes in itself a rigid frame exactly corresponding to that of a bicycle wheel. A slight mechanical improvement may be effected by placing rings of channel-section between the angles, and riveting the three rings together so that the whole frame is of metal.

The buckets and shrouding are attached to the two angle rings in the manner clearly shown in the detail—Figs. 129 and 130. The groove for the driving rope, which should be deep, is formed in the hard-wood segments above mentioned.

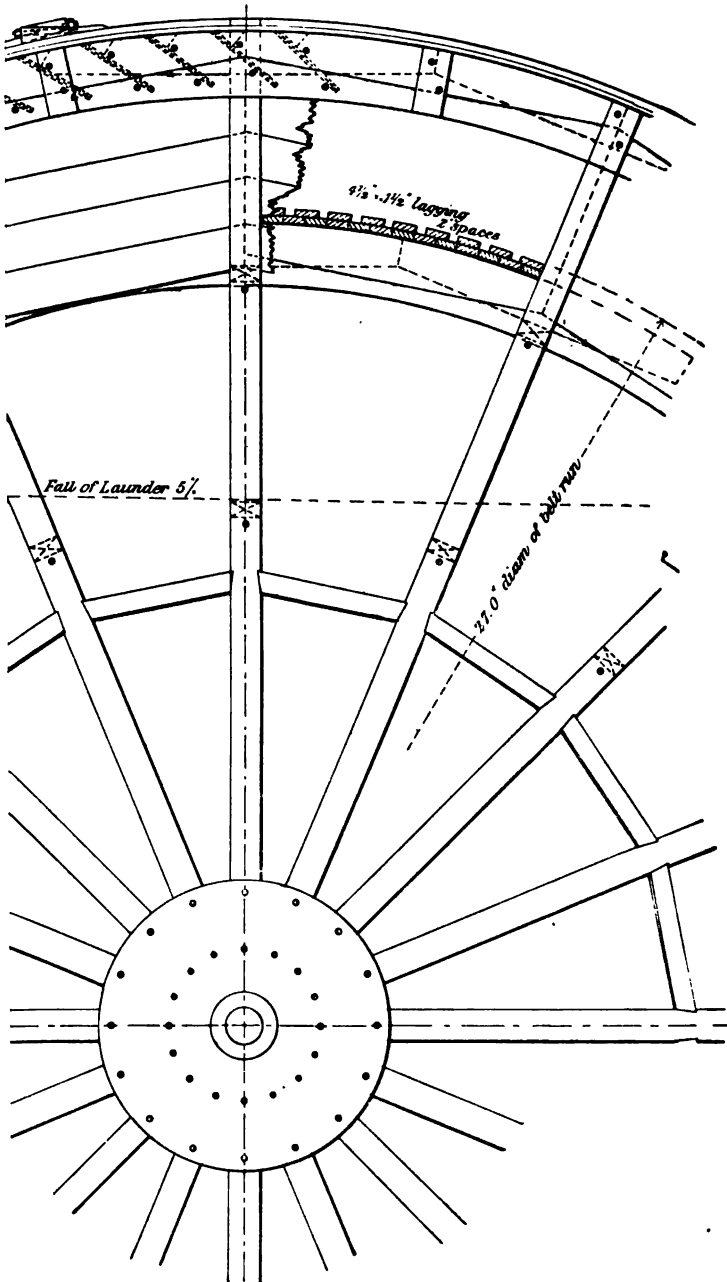
The following particulars of the 60-ft. tailing wheel at the Henry

<sup>1</sup> *Eng. and Min. Jour.*, vol. lxxvii, pp. 482-3, 1904.

Univ. of  
California



30



al Elevation.

Nourse mine, designed by L. I. Seymour, were supplied by F. J. Pollard, the company's engineer :

Diameter of driving pulley on wheel, 55 ft. ; axle, 20 ft. long by 14 in. in diam. ; bearings, 2 ft. 6 in. long by 12 in. in diam. The wheel and a top platform, 42 ft. by 7 ft., are carried on steel-framed towers. The wheel is driven by a  $1\frac{1}{2}$ -in. rope at 3 r.p.m. from a pulley 11 ft. in diameter, and takes 15 h.p. when lifting the pulp from 70 stamps. The efficiency is, therefore, about 45 per cent. The tension of the rope is maintained by a weighted carriage similar to those used for mechanical haulages.

H. L. Lavenstein has designed a spokeless wheel. The rim is much stronger and stiffer than that of the ordinary wheel, and is provided with a circular rail around each outer edge. The rails are supported on rollers driven by gearing, and these drive the wheel by friction. With this wheel the usual heavy framework for carrying the bearings is dispensed with, and the driving arrangements and foundations are less costly.

The tailing wheel illustrated in Figs. 131 and 132, from drawings kindly lent by Ernest Williams, of Johannesburg, is of the double type, that is, with buckets on both sides of the arms, and with a driving pulley in the middle. It is 33 ft. in diam., and is designed for a net lift of 24 ft. When running at 4 r.p.m. it is suitable for lifting the pulp from a 100-stamp mill, crushing, say, 500 tons per day. It has 96 buckets on each side, laid out to a tangent circle 24 ft. 6 in. in diam., so that they make an angle of about  $50^\circ$  with the radius.



## CHAPTER XXXVIII.

### SLIME SETTLERS AND AGITATORS.

**The Robinson Slime Plant** forms a good example of the type used for the treatment of slimes, by the method of transfer by pumping, introduced by J. R. Williams, who kindly supplied the drawings from which Figs. 133 to 142 have been reproduced. The plant was designed in the Eckstein engineering department, by A. M. Robeson, consulting mechanical engineer.

The chief feature of this plant is the simplicity of its general arrangement, as shown in Fig. 133, whereby the supervision of its working is greatly facilitated.

There are four collecting tanks in one row on raised foundations, and six other tanks in two parallel rows at a level 11 ft. below that of the collectors. Five of these are treatment tanks and one is used for solution. Each tank is 50 ft. in diameter with a 12-ft. vertical side and a conical bottom 3 ft. 9 in. deeper at the centre. The collecting and treatment tanks have a 10-in. central discharge opening (see Fig. 142) admitting the discharge pipe S, which is connected to the tank by a cast ring which also forms a seat for the valve G, when the outlet is closed. The valve G is attached, as shown, to a vertical flushing pipe F, suspended by a forked link D from the staging above. This link can be raised and lowered by turning the handle and nut E. The valve carries tapered feathers to ensure its true seating when the pipe is lowered. The top end of the pipe terminates in a bend F for attaching a hose, thus making a connection with the 6-in. solution mains, and allowing a stream of solution to be sent directly into the discharge pipe. An air pipe R is provided inside the pipe F for the aeration of the pulp. Each of the collectors has a steel, peripheral-overflow trough which delivers into a launder leading to the mill storage tanks. Light angle-steel towers (see Fig. 136) are fixed in the middle of the tanks to carry the vertical pipes and also to support the ends of the gangways leading to them.

Each collecting and treatment tank has a 5-in. hinged decanter pipe, which is raised and lowered by a small winch placed on the steel tower.

The staging giving access to the valves is of timber, and includes two main gangways extending lengthwise between the rows of tanks, with small cross-gangways to the centre towers and two other cross-gangways connecting the longitudinal ones.

The general arrangement of machinery and piping in the pump-house is shown in Figs. 139 to 142.







**DETAIL OF  
DECANTOR CONNECTIONS.**

DETAIL OF  
DECANTOR CONNECTIONS.

**MI  
S.**



**FIG. 138.**





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A 3/4

Recharge Valve





**Settlers with Mechanical Aids to Slime Removal.**—These may be divided into two sections according as to whether the action is intermittent or continuous.

*Intermittent.*—As the precursor to the continuous thickener this type is of interest. The vats used are somewhat similar to the slime thickeners at present used on the Rand, and with bottoms sloping slightly to a central discharge. The pulp is run in series through a number of these, being delivered centrally in each case, the overflow being distributed by means of an annular ring at the edges of the vat. As soon as one is filled with slime,

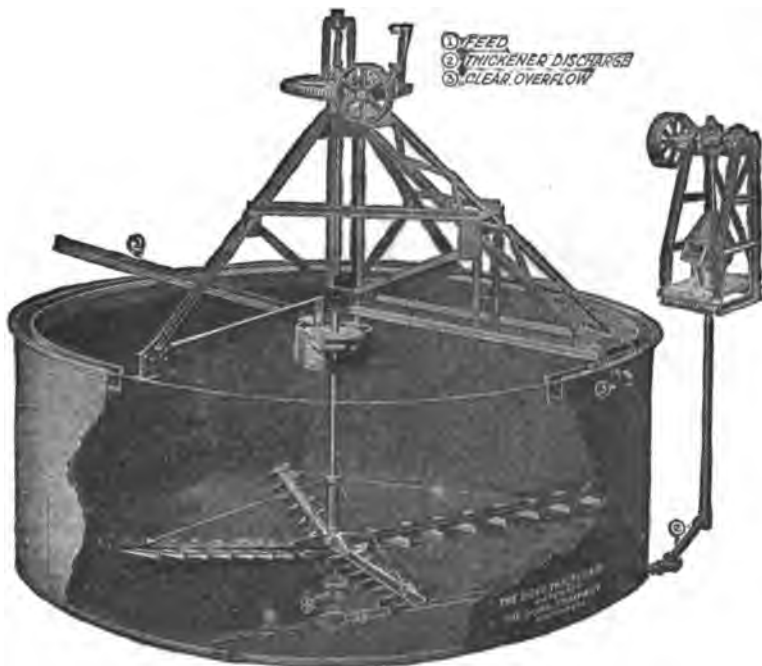


FIG. 143.—Dorr Continuous Thickener.

the stream of pulp is diverted and the contents allowed to settle. After a time the clear water is siphoned off, joining the overflow from the last vat, and a slowly moving agitator, at the end of a square vertical shaft and driven by a crown wheel through which the shaft is free to move, is gradually lowered into the slime until within a few inches of the bottom. The discharge gate is then opened and the thick pulp is shovelled into a vortex mixer to be thinned down with strong cyanide solution preparatory to transference to the agitator vats.

*Continuous.*—The Dorr thickener, which may be taken as the only machine of its kind in universal use, consists of a series of slowly-moving ploughs attached to a vertical shaft in the manner shown in the illustration (Fig. 143). These ploughs operate so that the settled slime is slowly dragged

towards the centre of the vat bottom, where underflow discharge arrangements are provided. The mechanism is driven by means of worm gearing at the top of the vat, and no footstep bearing for the vertical shaft is necessary. The machine may be arranged so that the ploughs may be raised during temporary shut-downs, and gradually lowered into the slime after the resumption of operations. The thickened underflow may be discharged directly underneath the tank, the rate of outflow being preferably controlled by a Bunsen valve; or the outlet may be connected with a diaphragm pump, placed slightly above the level of intake into the thickener.

The Dorr Tray thickener (Fig. 144) is a modification of the standard machine, in which a considerable increase in the settling area is obtained by

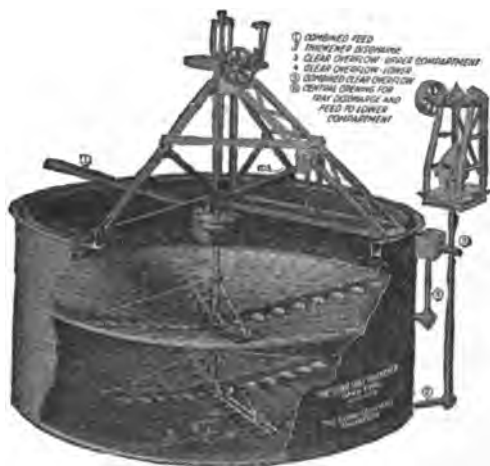


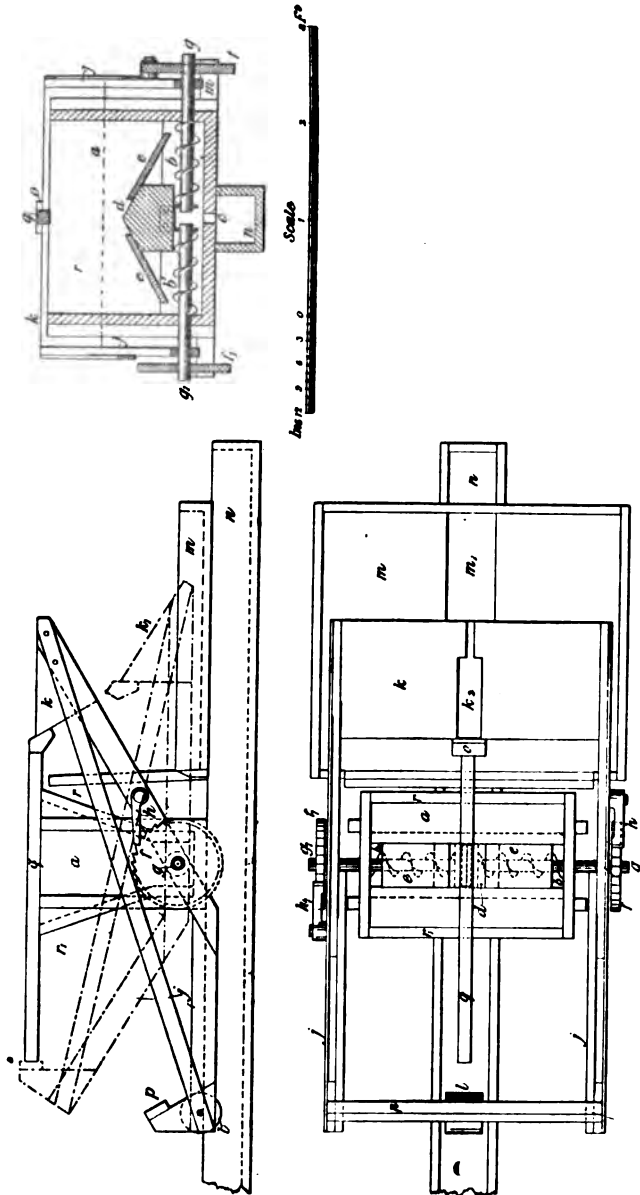
FIG. 144.—Open-Type Tray Thickener.

dividing the tank into horizontal compartments, each of which is fitted with ploughs and a floor. The latter conforms, in inclination, to the slope of the revolving arms.

**Feeding the Lime.**—If there are objections against feeding the lime into the battery with the ore, a small dry-crushing ball-mill may be used for pulverising this material. By raising or lowering the load of balls in the mill the rate of delivery can easily be adjusted to suit the feed, which is regulated by a bumper type of feeder capable of delicate adjustment.

An automatic apparatus for lime feeding, which only requires filling at long intervals, has been designed by P. S. Tavener, and is illustrated in Figs. 145, 146, 147. This machine has a wedge-shaped hopper *a* which is filled with powdered lime. The hopper has inside it two helical threads *bb*, at the bottom (on separate shafts *gg*), which push the lime from each side towards a central discharge hole *c*. This hole and the inner ends of the threads are covered by the block *d* and the inclined boards, *ee*, so that by adjusting the position of these boards on the block the supply of lime to the threads *bb*.

can be roughly regulated. Ratchet wheels  $ff_1$  are fixed to the shafts  $gg_1$  and are actuated by pawls  $hh_1$  attached to a rocking frame  $i$ , which is loosely



FIGS. 145, 146, 147.—Tavener's Automatic Lime Feeder, as used at the Bonanza Mine, Johannesburg.

centred on the projecting ends of the shafts  $gg_1$ . A water box  $k$  is attached to one end of the rocking frame, and a counterweight  $l$  at the other end. The action of the apparatus is as follows, assuming that the rock frame is in the position shown by full lines in Fig. 145.

A small stream of water falls into the opening  $k_2$  of the water box, which, when sufficiently full, falls down on to the tray  $m$  as shown at  $k_1$ , thus moving the rocking frame into the position shown by dotted lines. During this motion the pawl  $h_1$  pushes round ratchet wheel  $f_1$ , the shaft  $g_1$ , and thread  $b_1$ . The water in the box  $k$  escapes through opening  $k_3$  into tray  $m$ , and thence through hole  $m_1$  into the lime water launder  $n$ . When the box  $k$  is empty the counterweight brings back the rocking frame to its first position, and during this movement wheel  $f$ , shaft  $g$ , and thread  $b$  are actuated. Thus at each movement of the rocking frame a small quantity of lime is fed towards the discharge opening  $c$  and falls into the launder  $n$ , whence it is washed into the slime launder at any convenient point by the water which is used to work the apparatus.

The block  $o$  on the water box, and the bar  $p$  at the other end of the frame, alternately strike the rod  $q$  at the end of each movement, and this rod is attached to the two inclined sides  $rr_1$  of the lime hopper, which are hinged so as to be capable of a limited motion. The movement of these hinged sides each time the rod is struck prevents the lime from packing, and shakes it down to the openings at the lower edges of the inclined boards  $ee$  over the outer ends of the threads  $b b_1$ . Thus the action of the apparatus is entirely automatic, and it has proved successful in practice. The feed can be further regulated by altering the supply of water to box  $k$ , or by altering the size of its outlet at the narrow end.

**The Agitation of Slime Pulp.**—Methods of agitation of slime may be conveniently classified according to the means adopted for keeping the solid particles in suspension.

- (1) The direct application of compressed air, admitted at the bottom of the tank and allowed to bubble up through the pulp.
- (2) The use of compressed air in an air lift for the continuous elevation and agitation of the pulp, as in the Pachuca or Brown agitator.
- (3) Agitation by jet of solution supplied by centrifugal pump.
- (4) Mechanical stirrers, generally in the form of horizontal revolving arms fixed on a vertical shaft.
- (5) Combinations of these methods.

**Agitation by Compressed Air.**—The slimed-ore agitation vats in use at the Kalgurli mine form a good example of the direct use of air for such purpose. According to Robert Allen,<sup>1</sup> these vats, of which there are twenty in operation in this plant, are "each 12 ft. 6 in. deep and 6 ft. in diameter, the vat being cylindrical for a depth of 7 ft. 6 in., then a truncated cone for a depth of 5 ft., the bottom being flat and 18 in. in diameter. Two 1-in. air pipes, with  $\frac{1}{4}$ -in. valves, admitting a small supply of air for agitation purposes and diametrically opposite, follow the sides of the vat to the bottom, along which they run, terminating near the centre at a distance of about 6 in. from each other." An ingenious piping arrangement for discharge prevents settlement in the pipe and obviates chokage.

<sup>1</sup> *West Australian Metallurgical Practice*, p. 58.

**Pachuca Vat.**—The commonest type of vat using air for the purpose of agitation is the Brown agitator, or Pachuca vat (Fig. 148), which was first introduced in New Zealand and has since been used largely in Mexico for the treatment of silver ore.

It consists of a tall conical-bottomed tank having a central air lift in diameter about equal to one-twelfth the diameter of the vat. The top of the circulating column is just on or below the level of the pulp in the vat, so that the lift works at maximum efficiency, the depth of submergence equalling or exceeding the height of lift. The necessary air connections for the working of the system consist of an air connection for working the lift, an air connection for agitating pulp in the vat proper, and an auxiliary flexible connection. The best arrangement for the lift connection is to lay the air pipe inside the lift-pipe so that the air is discharged downwards, within a few inches of the bottom of the lift column. If this precaution is adopted there is little danger of chokage of the air pipe at any time, and the use of the bicycle valve or ball valve may be avoided at the discharge end of the pipe. This arrangement also obviates any necessity for bends or elbows being used in the air pipe after it enters the pulp, such connections being contributory causes to chokage in many instances.

The second air pipe should be laid outside the air-lift column and should terminate on a level with its lower extremity. This pipe, being also in a downward direction and containing no bends or elbows, is little liable to choke at any time. It is used for the purpose of agitating the pulp during intermittent charging or discharging, or to free the air-lift intake after a shut-down.

The auxiliary connection consists of a length of pipe attached to armoured hose, and thence to the air main, and of such a length that the air may be communicated to any part of the vat or system of vats. This auxiliary service is seldom necessary during ordinary working, but is of value in instances where coarse sand may occasionally find its way to the agitating department. A spider of air pipes, radiating from a central collar about on a level with the top of the cone, is of no practical value. In the event of chokage of one or more of the air pipes the pressure cannot be raised in any individual pipe to an extent which will dislodge an accumulation of slime. It is imperative that each outlet in an agitation vat should have its own valve connection with the pressure main.

**Single Vat Treatment.**—When the Pachuca vat was first introduced, each charge of slime was treated in a single vat. The charge was circulated by the air lift for the time required to dissolve the gold and silver, and was then drawn off through the outlet near the bottom of the cone and delivered to the feed tanks of the vacuum filter plant or to filter presses.

**Pachuca Vats in Series.**—In 1908, J. L. Mennell proposed to run pulp successively through a series of Pachuca vats. In 1910, H. M. Kuryla, at the Esperanza Mill, El Oro, connected in series a number of vats previously worked separately,<sup>1</sup> and the series method is now almost always adopted.

<sup>1</sup> "Continuous Agitation System," *E. and M. J.*, July 1910.

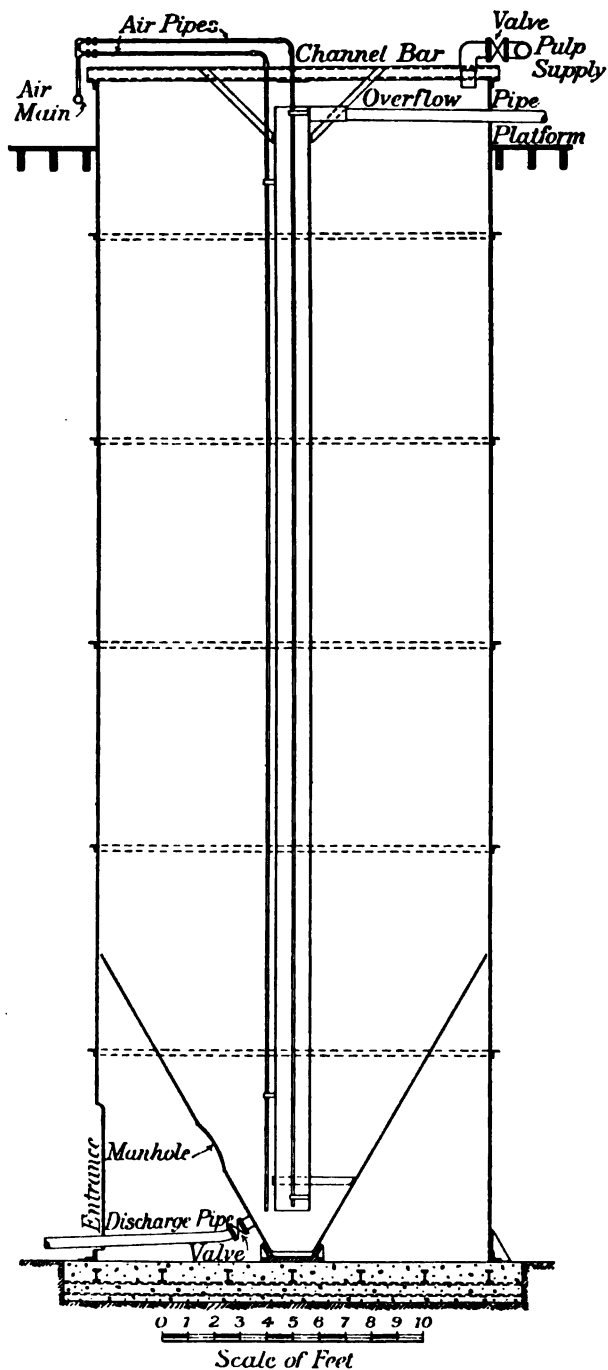


FIG. 148.—Brown Agitator or Pachuca Vat. Section through centre.

As compared with the previous method of treatment in the same tanks the gain in extraction is stated at 1·3 per cent. gold and 1·5 per cent. silver with a concurrent saving of 0·05 lb. of cyanide per short ton. This is chiefly due to the fact that with the series system the time available for agitation is increased by about 15 per cent.

Advantages of the series system have been pointed out by Huntington Adams.<sup>1</sup> The series connections at the Natividad mill, Mexico, were made in the following manner :

A drop of 4 in. is allowed from vat to vat, each air lift being 4 in. shorter than the preceding one. Around the top of each lift tube, a wooden overflow box is fixed, the top being level with that of the tube. The box is 6 in. deep, and from it a 4-in. pipe delivers to the next vat of the series. A 4-in. by-pass branch, provided with a valve, leads to the next vat but one, so that any vat can be cut out of the series. When the discharge from the last vat is taken from the top of the lift a considerable saving of head is effected.

In taking advantage of this saving of head in the adoption of a continuous system of agitation, it must be remembered that the pulp is delivered from the last tank in a continuous stream as received in the first : and that, unless a continuous filter is to be used, separate arrangements must be made for temporary storage of the pulp at times when the filter is not receiving its charge.

**The Parral Vat.**—At the Veta Colorado mill, Mexico, B. Macdonald<sup>2</sup> installed cylindrical, flat-bottomed vats 25 ft. diameter by 42 ft. high. Each contained four 12-in. diameter air lifts set 12 in. clear of the bottom, 4 ft. from the side of the tank and equidistant from each other. The air-lift deliveries are set tangentially, so as to produce rotation of the pulp in the vat. This arrangement might prove useful for the conversion of existing flat-bottomed tanks into air-lift agitators, but otherwise offers no apparent advantage over the Pachuca.

**The Paterson Hydraulic Agitator.**—This agitator (Fig. 149) is of interest as the forerunner of various schemes of agitation by decanted solution which have been introduced during recent years. In outward appearance the agitator much resembles the Brown or Pachuca vat, but agitation is effected by means of a centrifugal pump whose suction takes a continuous supply of decanted solution from a calm zone at the top of the tank. This decanted solution may be clarified and precipitated of its gold or silver contents *en route*, so that agitation may be accompanied by replacement of pregnant with barren solution.

Continuous agitation may be effected, as with the Brown agitator, by cutting off a portion of the lift discharge and delivering this into a second vat, and so on. A method of continuous agitation which does not ensure the pulp making at least one complete cycle of agitation in the vat is not to be recommended ; neither are ordinary cocks or valves suitable for the purpose of regulating the flow of pulp from one vat to another. All pipes used for

<sup>1</sup> *American Inst. Min. Eng.*, August 1911.

<sup>2</sup> *Trans. Am. Inst. Min. Eng.*, 1912.



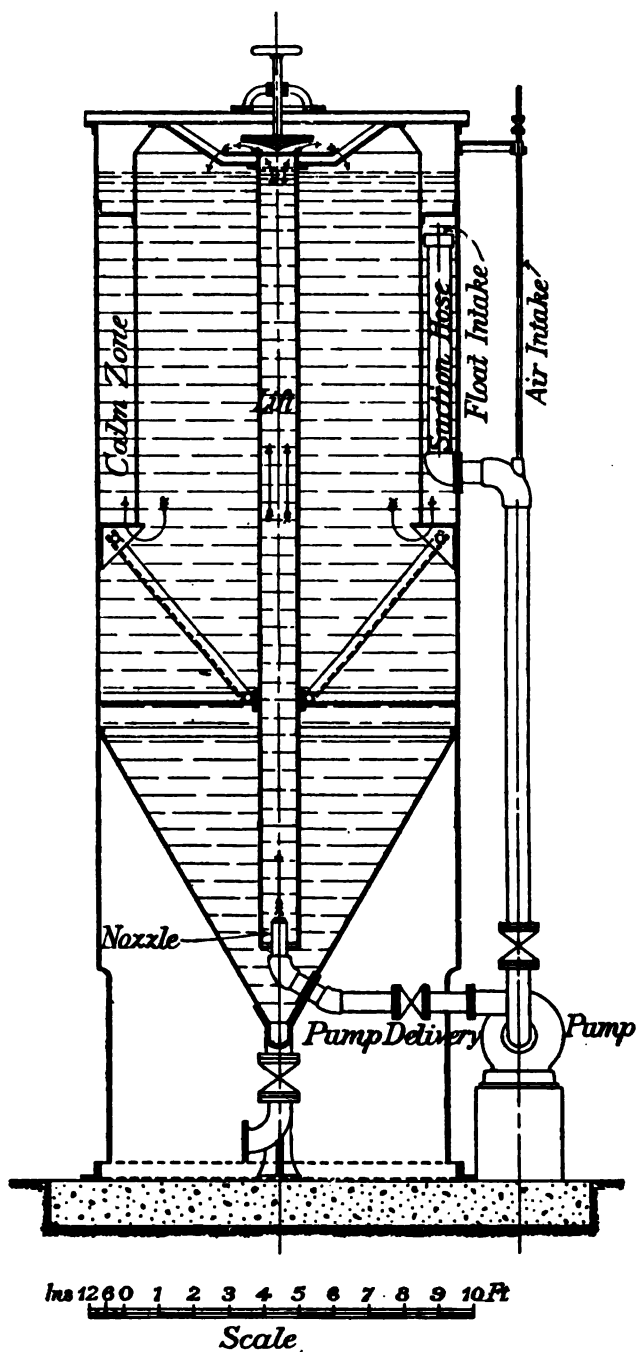


FIG. 149.—Paterson Agitator.

the conveyance of pulp which has little pressure behind it should be laid in such a manner that they may be rapidly and easily cleaned of any obstruction.

**Dorr Agitator.**—The Dorr agitator (Fig. 149A) consists of a vertical air lift which is slowly revolved by a shaft supported from the top of the vat in which it is set, and which is equipped with (1) distributing launders above the level of the liquid in the vat, and (2) plough arms at the bottom which sweep the solids which settle out from the pulp towards the centre, whence



FIG. 149A.—Dorr Agitator.

they are raised by the air lift and distributed by the launders. The agitator can be operated either continuously or intermittently, and has the advantage of being adaptable for pulp containing comparatively coarse sand—up to 30 mesh. The distributor arms are operated at speeds varying from 1 to 4 r.p.m., and means are provided for raising the plough arms so that the mechanism can be started without trouble after a shut-down. The power required to revolve the agitator mechanism varies from  $\frac{1}{2}$  h.p. for the small-size machines, up to 3 h.p. for the largest ones. From 8 to 30 cubic feet of air is required per minute, according to the size of the agitator.

**Mechanical Agitation Apparatus.**—The sketches Figs. 150 and 151 give a general idea of the arrangement of stirring gear formerly used on the Rand. A timber platform *a*, supported on trestle frames *b*, is fixed over the line of

vats, and carries a series of pedestals, *cc*, with bearings for a horizontal countershaft *d*. The platform also supports vertical bearings *ee* for the paddle shafts *ff*. The latter are sometimes hung on these bearings by collars, and in other cases are provided at their lower ends with footstep bearings on the floor of the vat. The paddle shafts are operated by bevel wheels *g g*, driven by pinions *h h*, each of which has a clutch to throw it in

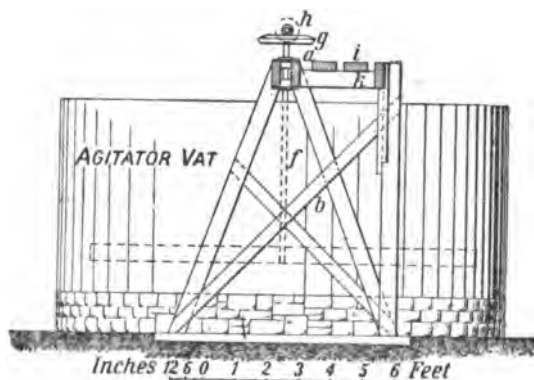


FIG. 150.—Stirring Gear: end elevation.

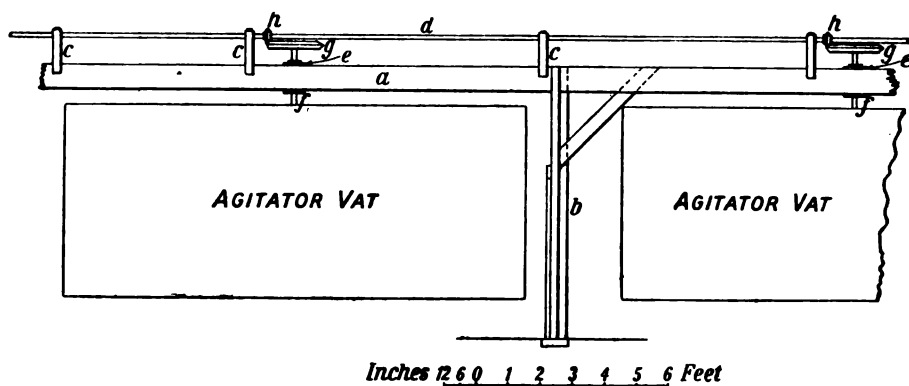


FIG. 151.—Stirring Gear: longitudinal elevation.

and out of gear with its corresponding wheel. A gangway *i* is provided for the shiftmen to attend to these clutches and for oiling. The whole platform is well braced against vibration along its full length by diagonal struts *k*, placed in a horizontal position under the gangway.

Agitation vats are sometimes arranged in the same way as the well-known Wheeler pan, *i.e.* the bevel gearing is below the vat, and the paddle shaft passes up through a central tubular shield which forms an integral part of the vat. The paddles are attached near the top of the shaft, and pass down outside the central tube into the pulp.

**Foundations for Slime Vats.**—When the coned bottoms of these vats are at or below the natural surface of the ground, this is cut out to fit the cone; tunnels are cut to give access to the centre, and are lined with masonry. The vat is bedded upon a layer of asphalt, or sometimes a layer of pitch and tar only. When the vats are to be raised above the ground level, the foundation is formed by building retaining walls round the area to be occupied by the vats and on each side of the passages under them. The spaces enclosed by these walls are filled in with dump rock, shaped on top to fit the vats, and finished off with a 3-in. layer of asphalt.

**Combination of Paddles and Air Lifts.**—At the Tonopah Belmont mill the agitation vats, 28 ft. diameter by 20 feet deep, are fitted with revolving paddles and also with an air lift.<sup>1</sup> The latter consists of a 6-in. pipe, fixed outside the vat, with a  $\frac{3}{4}$ -in. air pipe supplied at 15 lb. pressure. It draws pulp from the centre of the bottom and delivers it at the top at a point near the side of the vat.

At the Tonopah Mining Company's mill<sup>2</sup> similar vats are fitted with four air lifts to each. Each lift has a 10-in. outer box with a  $\frac{1}{2}$ -in. air pipe supplied at 20 lb. pressure. Two of the lifts deliver at 2 ft. from the side, one at 7 ft. from the side and one at the centre, so that the lifted pulp is fairly evenly distributed over the surface of the mass. The power consumed for a vat 30 ft. diameter and 20 ft. deep is stated at 6 h.p. for the air and 4 h.p. for the paddles when rotating at 5 r.p.m.

**The A.Z. Agitator**, as used at Waihi, New Zealand, consists of a screw propeller 5 ft. diameter over ends of blades, mounted on a vertical shaft in a tank 20 ft. diameter.<sup>3</sup> The effect of this is to produce an ascending column in the centre of the vat and the consequent descent of pulp at the side.

<sup>1</sup> C. T. Rice, *Eng. and Min. Jour.*, July 15, 1911.

<sup>2</sup> *Ibid.*, June 17, 1911.

<sup>3</sup> *Australian Min. Standard*, Sept. 15 and Oct. 20, 1909.

## CHAPTER XXXIX.

### VACUUM FILTERS AND THEIR OPERATION.

**Movable Leaf Vacuum Filter. Moore Type.**—This filter was first introduced at the Consolidated Mercur Mines, Utah, in 1902 by G. Moore, to whom belongs the credit of the successful application of vacuum filtration. His apparatus consists of a number of wooden frames, over which a canvas bag is stretched, so that each frame constitutes a vacuum chamber, and is known as a filter leaf. The canvas sides are kept apart by internal strips of wood which form part of the frame. A suction pipe passes through the top of each frame and extends downwards to about  $\frac{1}{2}$  in. from the bottom. Two blowing-in pipes also pass just through the top of the frame. A number of such frames are hung vertically, about 4 to 6 in. apart and parallel to each other, from a pair of channel bars, a group of leaves thus assembled being termed a basket. This constitutes one filtering unit which may contain from 2000 to 3000 sq. ft. of surface. A vacuum pump is sometimes fixed to the top of the basket and connected by a cross header to the suction pipes from the individual leaves. The basket is suspended from a crane or crawl for rapid transfer from one compartment of the filtering tank to another.

The filtering tank has three compartments, containing thick slime pulp, wash solution, and wash water, respectively. Beyond the third compartment, provision is made for discharging the residual slime.

In operating the plant, the basket is first lowered into the slime compartment, and the vacuum pump is started to suck solution through the canvas, thus forming an adherent cake of slime on the outside of each leaf. When the cake is thick enough, the basket is transferred by the crane to the weak solution compartment, the vacuum being maintained during removal. Here barren solution displaces the residual rich solution in the cake. The basket is then removed to the last compartment, where the weak solution is displaced by water. Lastly the basket is moved to discharge position and pressure air is admitted to the inside of the leaves to dislodge the cakes.

At the Victoria mill of the Waihi G.M. Co., a plant of this type has been in operation since 1908 and treats about 16,000 tons per month. These and the following details are abstracted from a paper by W. Macdonald :<sup>1</sup>

<sup>1</sup> "Vacuum Filtration at the Victoria Mill, New Zealand," *Jour. Chem. Met. and Min. Soc. of S. Africa*, May 1913.

## Time of cycle of operations :

|  |                |
|--|----------------|
| Taking on cake, . . . . .              | 50 min.        |
| Washing, . . . . .                     | 60 „           |
| Discharging, . . . . .                 | 24 „           |
| Transferring (3 operations), . . . . . | 10 „           |
|  | <hr/> 144 min. |

Tons per charge, 7·304.

Efficiency of displacement, 99 per cent.

Cost per ton of dry slime, 4·98 pence (10 cents).

The following figures given by G. Moore<sup>1</sup> are also useful. When working with a basket of 18 leaves, 20 ft. long by 4 ft. deep, thus aggregating 2880 sq. ft. of filtering surface, an 8 × 7-in. Deane pump gave the required vacuum of from 18 to 20 in., and each basket treated about 75 tons of dry slime in 24 hours. A 75-h.p. motor operated the crane, but as this was only in use for about 2 minutes per hour, it is equal to only 2·5 h.p. working continuously.

**Fixed Leaf Vacuum Filter (Butters and Cassel Type).**—Of the various modifications of filters on the Moore principle, the Butters' type is perhaps the most widely adopted. In this type the leaves remain in the filter box, which is successively filled with slime pulp, wash solution, and wash water. The filter boxes are generally of sheet steel, constructed in groups of three or more and with vertical sides. The portion below the leaves is hopper-shaped for convenience in discharging the residues, and the outlet at the bottom is closed by a large gate valve. A light overhead crawl is provided for handling the leaves singly, when repairs or cleansing are needed. In large plants all valves over 4 in. diam. are operated hydraulically from one central point, where the control valves and the vacuum and pressure gauges are mounted in the manner of an electrical switchboard.

The inlets of the boxes are connected to the pulp storage tank, the barren solution storage tank, and wash-water supply through one large centrifugal pump which handles each liquid in turn.

**The Butters' Filter Leaf.**—The latest pattern (see Fig. 152) is built up on a complete rectangular frame  $AA_1$  formed of  $\frac{1}{2}$ -in. piping with brazed joints. This is attached to a top header bar  $A_2$  of wood, whose ends rest upon the sides of the filter box when in use. The top and bottom pipes are perforated with  $\frac{1}{4}$ -in. holes at 3-in. centres. The side tubes  $A$  and  $A_2$  project through the header bar for connection to the header pipes  $J$ . In the figure, this connection is shown on one side only at  $K$ . One side is connected to the top pipe only and the other side to the bottom pipe only. The lower corners of the frame are now formed with the pieces, for temporary connections when the leaf is being cleansed, which are closed with plugs when the leaf is in use.

This frame is covered with sheets of 10-oz. or 12-oz. canvas,  $CC$ , stretched over it and sewn together round the edges, coco matting,  $B$ , being

<sup>1</sup> *Eng. and Min. Jour.*, Dec. 5, 1903.

placed within to keep the sheets of canvas apart when under vacuum. In some cases, vertical wooden strips are used inside, instead of the matting. Vertical lines of stitching *F*, stiffening strips *G G*, and eye-bolts in the header bar complete the leaf. The standard size adopted is usually 10 ft. long by 5 ft. deep.

A main header pipe runs along each side of the filter box, each having tee branches which are connected to the leaves by a special fitting containing a glass sight tube and means for rapid connection and release.

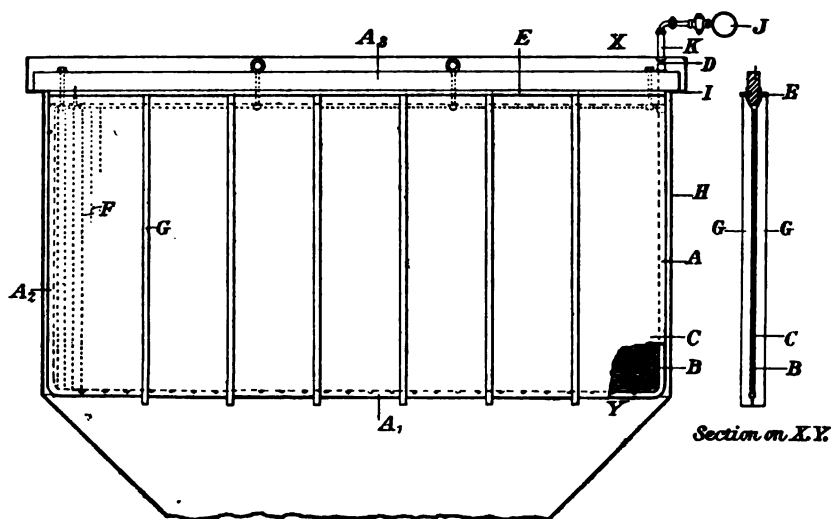


FIG. 152.—Butters' Filter Leaf. Elevation and Section.

**Butters' Filter Practice.**—The method of procedure varies slightly under different conditions. At the Butters' Copala Syndicate mill, where all-sliming in cyanide is adopted, the pulp at 3 to 1 dilution is treated in agitation vats and is then passed to a filter feed tank, where more solution may be decanted while awaiting filtration. When the final pulp has a specific gravity of 1.3 to 1.4, it gravitates through a 6-in. pipe to the filter box until the leaves are completely covered. Then the header pipes are put in communication with the vacuum drum, and solution is sucked through the canvas into the interior of the leaves and thence to the drum, from which it is pumped to the precipitation boxes. An adherent cake of plastic slime is thereby formed on the outside of the canvas. A low vacuum is used at first, about 6 to 8 in. of mercury; but, as the cake thickens, the vacuum is raised. As solution is thus withdrawn from the filter box, more pulp is added at intervals to keep the leaves submerged. This causes a periodical upward flow which, in this particular case, provides sufficient agitation to prevent settling of the heavier slime particles.

Generally, however, a pulp of greater dilution than 1 to 1 requires continuous circulation in the filter box and this is effected by a small outside

air lift. On the other hand, pulp which contains less than 1 of water to 1 of slime is usually not sufficiently mobile for economical handling, but there are exceptions to this rule. The coarser the slime particles the less solution need be present.

When the cakes are thick enough, in this case about 1 in., the vacuum is lowered to 5 or 6 in. and the surplus pulp in the filter box is pumped back to the pulp storage tank. The box is then filled with a weak cyanide solution and the maximum vacuum applied in order to wash out the residual gold-bearing solution. A water wash is similarly applied and the main vacuum pipe is then closed. Water at 8 to 10 lb. pressure is admitted to the interior of the leaves through the header pipes. This water oozes through the cloths and causes the cakes to slide off and fall into the hoppers, whence the slime and water flow to the residue dam.

**Blowing off Cake with Air.**—Compressed air at about 10 lb. pressure is used at some plants instead of water for dislodging the cakes from the leaves. Air alone is liable to blow off patches of the cake instead of removing it completely. It has been found that the cake separates better when a small quantity of water is used in conjunction with the air. The method usually adopted is to supply air through the bottom pipe and water through the top one in just sufficient quantity to saturate the coco matting. The water produces a slippery surface outside the canvas, and causes the cake to slide off in one piece. By continuing the air supply for a minute afterwards, the pores of the cloth are cleaned and ready for the next cycle.

Difficulty in discharging the cake by any of the above-mentioned methods is, however, a serious disadvantage in many installations.

**Filling and Emptying Filter Box.**—The time required for this depends upon the head available, and the size of pump and piping. In a small plant of, say, 30 leaves, the conditions generally admit of filling in four or five minutes, but with larger plants, neither the head nor the piping can be increased in proportion, so that with 150 to 300 leaves the time for each filling is from ten to fifteen minutes. In this respect the fixed leaf compares unfavourably with the movable type.

**Simultaneous Replacement of Pulp with Wash.**—The principal drawback to efficient working in leaf filters arises from the ill-effects of the exposure of the cake to the atmosphere between formation and wash. This often results in cracking, interference with molecular arrangement of slime particles, and imperfect displacement with wash. In order to obviate this trouble an interesting modification in ordinary fixed-leaf vacuum-filter practice was introduced, which was successfully adopted on various properties in Mexico. By this system<sup>1</sup> the excess pulp, after cake formation, was pumped from the lowest point of the filter box and was replaced at the same time with solution delivered by suitable distributors on top. The admixture of slime pulp and solution during the operation was found to be negligible and did not in any

<sup>1</sup> "Improvements in the Treatment of Slime by the Vacuum Filter Process," by A. W. Allen, *Eng. and Min. Jour.*, May 15, 1909.



way adversely affect the efficiency of the wash. Also, there was an obvious economy of time, which largely increased the capacity of the filter. Power costs were also reduced, and the non-exposure of the cake to the atmosphere resulted in improved conditions for displacement of pregnant solution with wash.

**Leaf Filter Operation.**—In the following details concerning the operation of leaf filters, which to some extent apply equally to fixed or movable types, the matter contained in a series of articles by H. Forbes Julian<sup>1</sup> has been freely used.

**Cake Formation and Thickness.**—When filtering slime containing much amorphous clay, it is good practice to apply a low vacuum (about one-quarter of the maximum) until the cake is about  $\frac{1}{8}$  in. thick. By this means a more permeable cake is formed than if the higher vacuum is applied at the start, and it is consequently more quickly washed. With crystalline material, such as a tube-mill product, the benefit of a low initial vacuum is not so apparent, but it usually results in a clearer filtrate. Operators as a rule, however, pay little attention to this point, but use constant vacuum throughout the cake formation.

At the beginning of each cycle, when the cloths are clean, the solution filters through readily, and, with a constant vacuum, the rate becomes slower as the deposit increases. A point is reached when it becomes more economical to stop formation, to wash the cake, and to start the cycle again. This point must be found by trial, because it varies with the nature of the slime. In some cases, when dealing with amorphous or gelatinous material, the limit is reached when the cake is only  $\frac{1}{2}$  in. thick, but with crystalline siliceous slime a cake of 2 in. or more may be economically formed.

**Permeability of Cake.**—The porosity of a cake is affected by the pressure of formation. Slime deposited by vacuum ranges from 40 to 65 lb. dry ore per cubic foot, whereas filter-press products contain from 65 to 100 lb. of solids per cubic foot.

G. Moore first pointed out the fact that, in the formation of cake by vacuum, if the resistance to percolation varies at different parts, a self-adjustment takes place, owing to the parts of greatest permeability taking on the deposit most quickly. The result is a cake, not of uniform thickness, but of uniform permeability, which is much more important.

**Washing the Cake.**—When the cake is formed it contains from 30 to 35 per cent. of gold-bearing solution, together with some undissolved gold or silver. The former can be removed by a water wash; but if any of the latter is to be recovered, an intermediate solution wash is necessary.

Various independent investigations have shown that some of the gold and silver which remain undissolved in the treatment tank may be dissolved by the solution wash. In view of the time occupied by and the cost of previous operations on the ore, it seldom pays to curtail solution washing. The extra metal obtained often far exceeds the cost of recovery.

<sup>1</sup> "Losses in Cyanidation," *The Mining Mag.*, Oct. 1911.

The following results at the Goldfield Consolidated mill given by J. W. Hutchinson<sup>1</sup> are sufficiently convincing on this point, as the figures were obtained at a time when the slime was receiving six hours more agitation per charge than in the normal treatment.

|  | Oz. Au. |
|--|---------|
| Samples of 91 charges from Pachucas to filters (thoroughly washed with water), . . . . . | 0.095   |
| Samples of 200 discharges. Same pulp, . . . . .  | 0.088   |
| Dissolved on filter, . . . . .   | 0.007   |

Equals 7 pence (14 cents) per ton of dry slime.

It was found at this mill that the displacement of the pregnant solution and the additional gold dissolved depend more upon the time of contact than upon the total quantity of wash solution. Hence a long wash of 85 to 100 minutes is given with reduced vacuum.

It follows therefore that, for the solution wash, the thickness of the cake makes little or no difference to the time required for the wash.

**Preparation of Wash Solution.**—Precipitated solution is generally used for washing. It should be thoroughly aerated; and if not already turbid, it is often thought advisable to add some fine slime for the readjustment of uneven permeability. Just before use, the cyanide strength should be increased above that of the treatment solution, especially when dealing with silver sulphides. It is generally inadvisable to add free alkali to wash solution.

**Water-washing the Cake.**—At this stage the cake contains about  $\frac{1}{2}$  ton of solution per ton of ore, and this solution carries an appreciable quantity of cyanide and some dissolved gold, so that a water wash is generally economical.

Fig. 153 illustrates the manner in which one wash actually displaces another as shown by experiment. ABCD is an ideal curve showing what would happen if the displacement were absolutely perfect, and serves as a standard of comparison for the experimental curves. In this ideal case the valuable contents of the effluent (Au or CN) would be constant, as shown by line AB, until the whole of the original moisture had been expelled. When the incoming wash reached the inner face of the cake there would be an instantaneous drop BC to the valuable content of the wash liquor, i.e. zero in the case of water. CD would represent the remainder of the washing process.

Under these circumstances, the time required for washing would be directly proportional to the thickness of the cake. Curve AEFD, Fig. 153, is plotted from figures given by J. W. Hutchinson<sup>2</sup> and the ordinates represent assay contents. Curve AGHD is that of H. F. Julian<sup>3</sup> for KCN strength in pounds per ton for the case of a water wash.

The two latter curves, which are remarkably alike in general shape, show by their initial drop that some dilution of the residual moisture occurs at

<sup>1</sup> "Operation of the Goldfield Consolidated Mill," *Min. and Sci. Press*, May 20, 1911.

<sup>2</sup> *Min. and Sci. Press*, May 20, 1911.

<sup>3</sup> *Min. Mag.*, Oct. 1911.

the beginning. Then follows a nearly horizontal line indicating a regular displacement. Then occurs a rapid drop EF and GH, corresponding with the ideal vertical line BC, followed by a final period of slow decrease in values. This last phase, FD and HD, probably represents the dilution and removal of some of the residual solution which has been held by absorption.

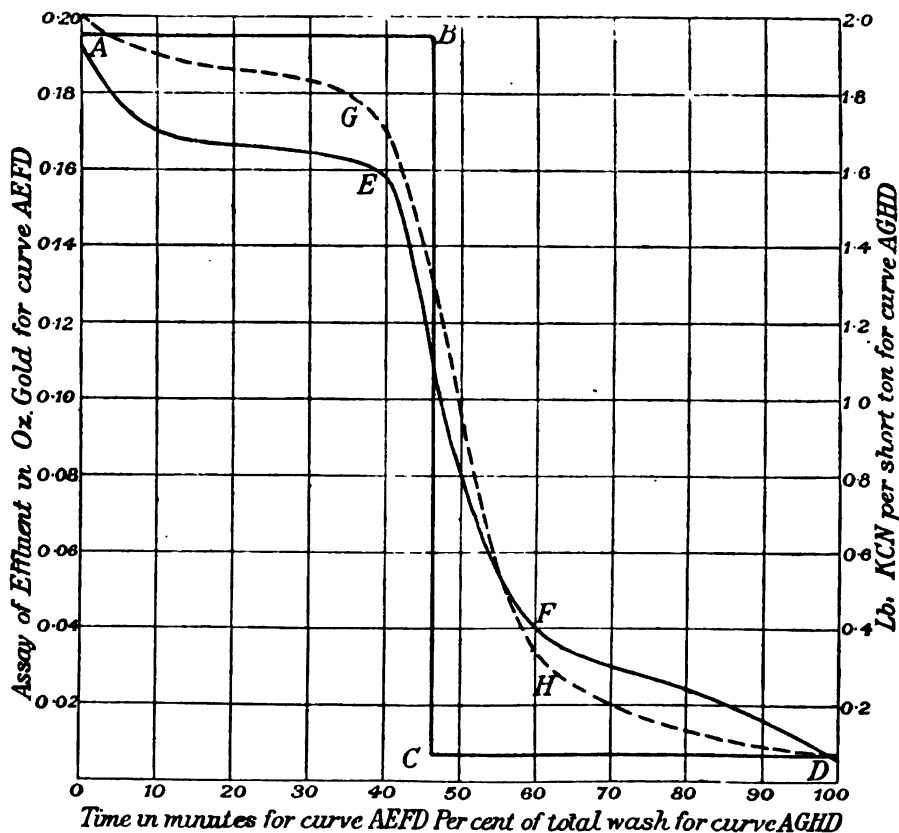


FIG. 153.

**Application of Leaf Filters to a Crushing Plant.**—This is shown diagrammatically in Fig. 154. The diagram, which refers to an all-sliming plant, deals with the handling and treatment of the ore from the stamp battery to a Butters' filter, and is practically self-explanatory. The pulp, as it leaves the battery, flows to two 8-ft. cones, with angles of  $60^\circ$  built in the hoppers of two tube mills. The underflow, which has a dilution of 1 to 1, goes through the tube mills; and, as it emerges, unites again with the stream of slime and solution from the overflow. The whole then passes on to four 32-in. cones with angles of  $65^\circ$ . The overflow from these small cones is the finished product (200-mesh), and passes on to the collecting tank. The underflow is pumped back to the 8-ft. cones. The proportion of 200-mesh

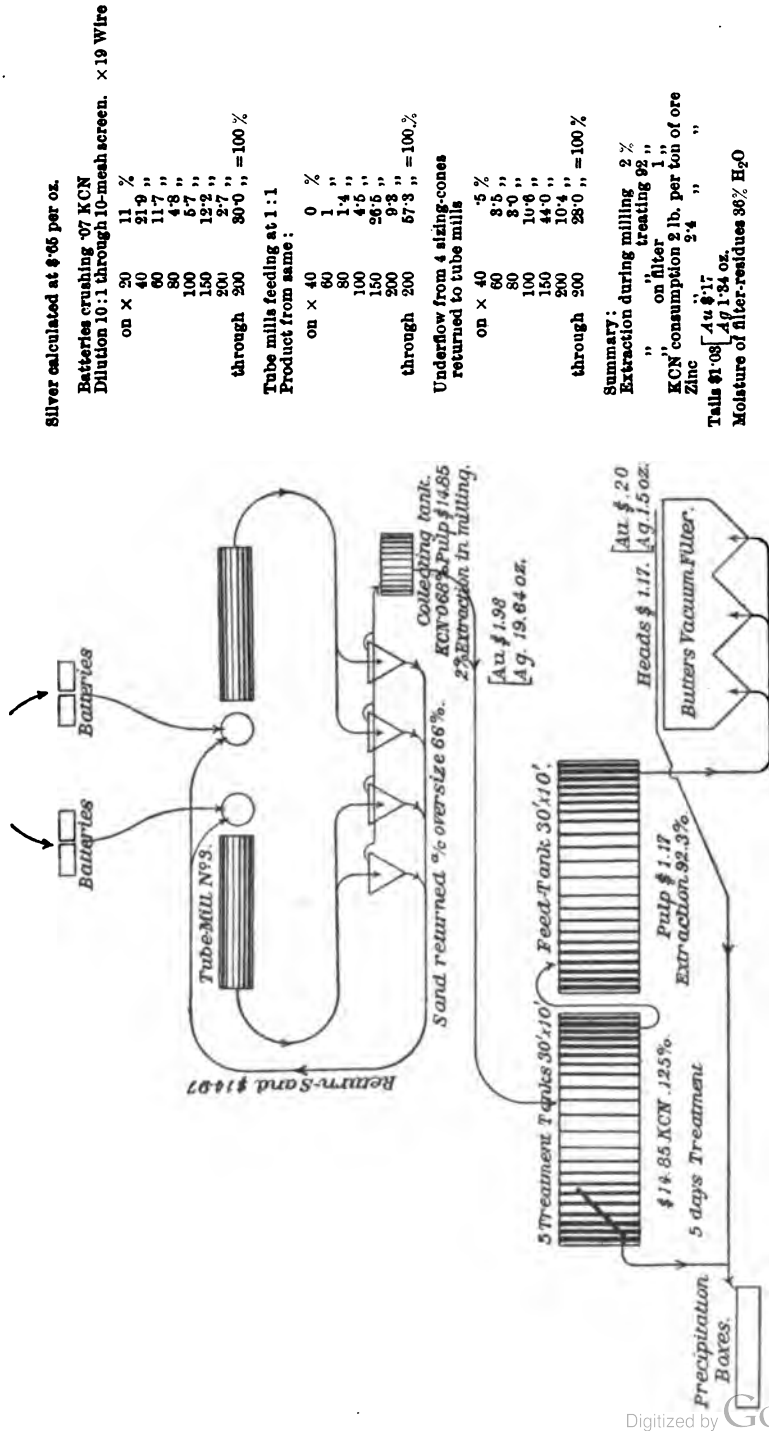


FIG. 154.—Butters' Copala Syndicate's Treatment Chart.

product passing away with the underflow being large, an upward current of solution is being provided in the cones.

**The Cassel Vacuum Filter.**—This is also an adaptation of the leaf filter, and involves various devices and operations patented by H. R. Cassel. The leaf, as described in the specification, is of similar construction to Moore's, but over the filtering surface hangs a loose protective cloth or mantle suspended from a movable frame capable of giving it a shaking motion. The leaves are kept stationary in the filter box during operations, and the pulp and water wash are brought to them, and removed in a manner somewhat similar to that described under Butters' leaf filter. The pulp, while in the filter box, is kept in a state of agitation, and on exhausting the space between the cloths of the leaves the pulp is said to adhere and gradually thicken against the mantles. When the cakes are sufficiently thick the surplus pulp is removed from the box and water is introduced in its place. This is sucked through the cakes until the dissolved metal held by them is displaced. The vacuum is now cut off and the surplus water is returned to its tank. At the bottom of the filter box, which is hopper-shaped, are doors held up by screw-jacks and capable of being supported on a wheeled carriage. On removing the doors, the frame supporting the mantles is caused to oscillate, the object being to give a shaking motion by which the adhering cakes fall off and pass through the doors, to be conveyed to the dump.

**Cleaning Filter Leaves.**—The pores of the canvas gradually fill up with lime compounds and consequently the leaves need cleaning at intervals. The first step after removal from the filter box, is to wash off all loose dirt and alkaline solution. The leaf may then be placed in dilute HCl (from 2 to 3 per cent.) for an hour or so. This removes the lime deposit, and after soaking in clean water the leaves are again ready for use. The use of a still more dilute acid (from  $\frac{1}{2}$  to 1 per cent.), is recommended in some instances, to be passed continually to and fro through the canvas until the deposit is removed, with subsequent washing by clean water introduced through the tubular frame. New filter leaves and all those not in actual use should be kept wet, preferably by submergence in clean cyanide solution, over 0.01 KCN and free from lime.<sup>1</sup>

**Drum Vacuum Filters.**—There are two well-known filters of this type, the Oliver and the Portland. In the first named, the drum is formed of wooden staves secured to a pair of spiders, keyed on to the driving shaft. Outside the drum is a cylinder of wire screening with distance pieces between it and the staves. The cylinder is covered with two layers of burlap and one of No. 12 duck. The space between canvas and drum is divided circumferentially into a number of separate compartments, usually twenty-four, which constitute the vacuum chambers. When in operation, these compartments are successively connected through the shaft to sources of vacuum and pressure, in due order, by the rotation of the drum. The drum is partly submerged in a bath of slime pulp, and a thin cake of slime continuously

<sup>1</sup> *Slime Filtration and Cyaniding*, H. F. Julian. O. Butters & Co., Ltd.

forms on the lower portion of the filter cloth. The adhering cake, as it leaves the pulp bath, is washed by means of sprays of solution or water, which are drawn through the cake by vacuum. The residue peels off at a point just before the cloth re-enters the pulp, and the discharge is assisted by the application of compressed air on the inside of the cloth.

**Horizontal Leaf Continuous Filters.**—The Ridgway filter has an annular trough divided into compartments for slime pulp, solution wash, water wash, and discharge hopper. Radiating from a central vertical shaft are a number of arms carrying small filter frames which successively dip into the pulp and wash compartments. As each filter frame passes over the discharge hopper it is tilted over and compressed air is admitted into it to ensure the dropping of the cake. The regulation of vacuum is entirely automatic.

**Table Filter.**—In a type of table filter used in India<sup>1</sup> the filters are stationary and the pulp and wash solutions are supplied by the revolving arms of a reaction wheel, similar to the Butters-Mein distributor. By making the central hopper of the distributor in several concentric sections, each supplying one arm, no valves are necessary in the rotating portion of the machine. This is a distinct mechanical advantage. The reaction wheel tilts each filter when it requires to be discharged.

Table filters are particularly applicable in the case of material which will not form an adherent cake upon a vertical leaf or upon the surface of a revolving drum.

<sup>1</sup> H. M. Leslie, *Jour. Soc. Chem. Ind.*, March 1908.

## CHAPTER XL.

### CONSTRUCTION AND OPERATION OF PRESSURE FILTERS.

PRESSURE filters are adopted to treat slime or slimed ore for a number of reasons. The use of the ordinary plate-and-frame type ensures a minimum loss in residue moisture, and has been adopted, as an imperative necessity, in regions where water is scarce or expensive. Other types are favoured because of the high pressure that is available to separate ore from solution, and to wash the cake which is formed. The filtering of many classes of colloidal material is only practicable in a filter press.

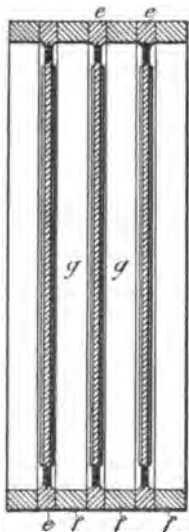


FIG. 155.—Plate-and-Frame Filter Press.

**Dehne Filter Press.**—This is the standard type of plate and distance frame filter press. Fig. 155 is a vertical section of a number of the plates and frames in such a press. The corrugated plates *e* and the separate distance frames *f* are placed alternately, thus enclosing the filtering chambers *g*. The filter cloths are hung over the plates. The channel or channels are formed either in projecting lugs or in the margins of the plates and frames. The cloths themselves constitute the packing between the planed margins of plates and frames. The joints in the channels in the lugs are made good by filter-cloth bags slipped over the lugs, or by cloth-covered india-rubber rings placed in a recess formed round the holes. Each plate is also usually provided with an efflux tap for the discharge of filtered liquor; and when it is not necessary to discharge under pressure, the use of these separate taps, instead of a discharge common to all the plates, is preferable. This enables the operator to see whether the filtration is satisfactory from each individual chamber, so that he can shut off any chamber which is not filtering properly.

Frame presses giving cakes 2 in. or 3 in. in thickness are generally employed in the treatment of slime, each press having usually 50 chambers with plates 40 in. square. Larger ones have been used, but are not recommended by the makers. A press of the size mentioned with 2-in. frames holds about 3 tons, and with 3-in. frames about 4½ tons, when full of pressed slime weighing 80 lb. per cube foot. As the 3-in. press costs only 12 per cent. more than the 2-inch, it is evidently cheaper to use the former whenever

the material will allow of this without unduly retarding the speed of filtration, because an economy is effected in first cost as well as in the work of emptying.

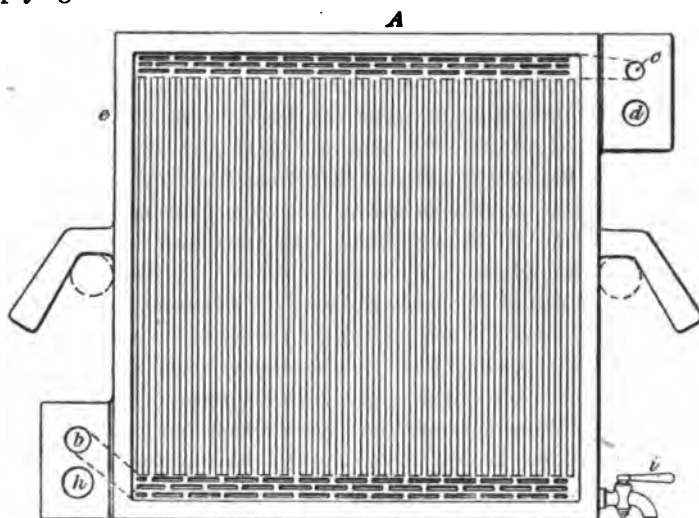


FIG. 156.—Dehne Press Plate.

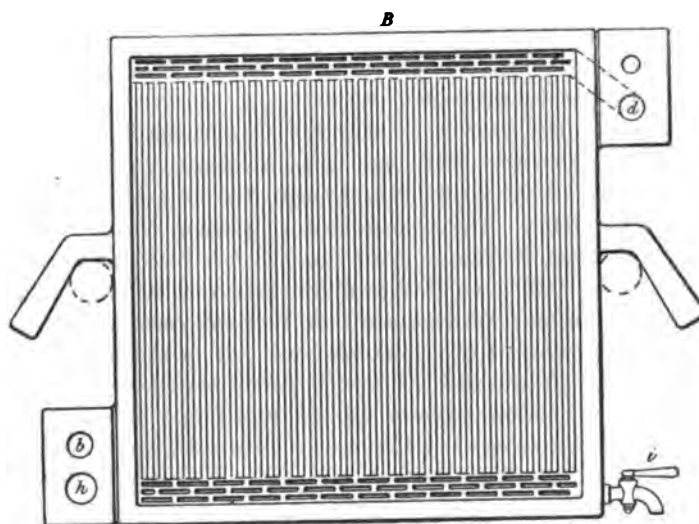


FIG. 157.—Dehne Press Plate.

In forcing the pulp into the chambers the pressure may be increased, often with advantage, as the cakes get thicker, but it should always be steady; therefore when pumps are used for this purpose, a large air vessel should be provided to prevent fluctuation.

In early practice, presses were usually filled by a montejus, which is



simply an air-tight vessel, usually cylindrical, with dished ends, provided with a valve at the top for filling it with pulp. It has also an inlet for compressed air at the top and a pulp outlet pipe, which extends nearly to the bottom of the montejus at its receiving end, and is connected at its other end to the feeding channel of the filter press. The capacity should be sufficient to hold the pulp required to fill one press with cake. For example, if the pulp contains  $1\frac{1}{2}$  parts of water to 1 part of slime, the montejus for a  $4\frac{1}{2}$ -ton press should contain 3.6 tons dry slime, at 12.2 cu. ft. per ton, = 44 cu. ft. plus 5.4 tons of solution at 32 cu. ft., = 173, or altogether 217 cu. ft. Under these conditions one montejus is necessary for two presses. Pressures from 45 to 100 lb. per square inch have been used for forcing the slime pulp into the presses.

**The Displacement of Valuable Solution** from residue in the operation of a chamber filter-press is generally effected by one of two means. Where all solid plates are low pressure, the outlets are always open to discharge and no drain cocks are necessary. The pulp is forced under pressure into the press and the solution filters through the cloths which hang between the solid plates and the hollow frames. The former have a channelled surface to facilitate the even flow of solution from the filtering area. Cake formation starts over the whole surface of the cloth and continues until the frame is completely filled with caked slime. A distinct line is traceable vertically through the centre of each cake. This line constitutes a zone of division which is noticeable when the presses are being discharged by hand. The displacement of residual solution carrying metal is effected by a wash operating through the same channels as those traversed by the pulp and charge solution. The wash solution or water enters the filling port in each frame, traverses the zone of division in each cake, and passes through the cake on each side of the centre line, through the filter cloth, and thence to the discharge launder. The same system of washing may follow the part-filling of the frame, so that a clear space is left between separate cakes formed on each cloth. In this instance, however, the advantages of a slightly higher washing efficiency are more than counterbalanced by the fact that satisfactory removal of the excess wash left in the hollow space is a difficult matter. Compressed air is only partly effective in reducing such moisture content because, however applied, it will find its way through the top of the cake where there is a minimum moisture content, to the avoidance of the lower portion where there is an excess. In a dry-discharging press this method results in a sloppy residue and, under any circumstances, an unnecessary loss of wash.

In the second method of operation the low-pressure plates alternate with high-pressure plates and the outlets from the latter are controlled by suitable valves. The low-pressure plates, as in the other system, are open to discharge at all times. During filling, all discharge valves are open, but as soon as the cake is formed the high-pressure valves are closed and wash is forced in, through a separate channel, to the high-pressure plates. This solution

finds its way through the cloths over the high-pressure plates and across the cake of slime, through the cloths covering the low-pressure plates, and thence to the discharge launders.

Given similar conditions of operation, it has been found that the efficiency by either method is dependent on the thickness of the frames in which the cakes are formed—that is, the thickness of the cakes themselves. The cross-washing system, with high-pressure plates, gives excellent results where the thickness of the cake is not too great. On the other hand, the median washing system is preferable where it is more economical or necessary to form thick cakes. As in all other classes of filters and filter-presses, the efficiency of displacement is largely affected by the thickness of the pulp being handled, rapid cake formation being the surest preventive of a segregation of coarser material in the lower portion of the frames.<sup>1</sup>

**Treatment Capacity of Presses.**—The average time for the cycle of operation of a Dehne filter press is about 60 min. per charge. In estimating the actual capacity of presses, however, it is necessary to allow for the time occupied in replacing filter cloths, in repairs, and for other incidental delays. The last column in Table LXVII. shows the press contents per ton of ore treated. These figures have been calculated from R. Allen's data,<sup>2</sup> and also from a paper by R. Godai.<sup>3</sup>

TABLE LXVII.—*Treatment Capacity of Presses in Practice.*

| Name of Mine.                    | Number of Presses. | Thickness of Cake. | Tons dry Slime per Press. | Total Contents of Presses. Tons dry Slime. | Averages tons treated per day. | Press Contents provided per ton treated per day. |
|----------------------------------|--------------------|--------------------|---------------------------|--|--------------------------------|--|
| Ivanhoe . . . . .                | 6                  | 2 in.              | 3·2                       | 29·8                                       | 292                            | 0·102  |
| Western Australia . . . . .      | 2                  | 3 "                | 5·3                       |  |                                |  |
| Kalgurli, W.A. . . . .           | 6                  | 3 "                | 4·75                      | 28·5                                       | 413                            | 0·069  |
| Gt. Boulder Prop., W.A. . . . .  | 4                  | 3 "                | 4·5                       |  |                                |  |
| S. Kalgurli, W.A. . . . .        | 9                  | 3 "                | 3·6                       | 50·4                                       | 514                            | 0·090  |
| Associated, W.A. . . . .         | 7                  | 3 "                | 4·3                       |  |                                |  |
| Lake View Consol., W.A. . . . .  | 7                  | 3 "                | 4·2                       | 29·4                                       | 335                            | 0·088  |
| Yamagano, Japan . . . . .        | 9                  | 3 "                | 5·1*                      |  |                                |  |
| New Goch, Johannesburg . . . . . | 2                  | 3 "                | 4·0                       | 8·0  | 64                             | 0·125  |
|                                  | 6                  | 3 "                | 3·87                      | 23·2                                       | 279                            | 0·083  |

\* Concentrate slime.

**Merrill Filter Press.**—This press, as described by Chas. H. Fulton,<sup>4</sup> differs from the ordinary slime press in being designed to discharge automatically the pressed cake by sluicing, without the necessity of disturbing the frames or plates and cloths. "The chambers of the press are filled from

<sup>1</sup> "Filter-Press Operation," by A. W. Allen, *Min. and Sci. Press*, April 15, 1914.

<sup>2</sup> *West Australian Metallurgical Practice*.

<sup>3</sup> *Proc. Inst. C.E.*, vol. clxxxiii.

<sup>4</sup> *Eng. and Min. Jour.*, vol. lxxxi, p. 76-77 f.

a continuous channel, running along at the median line at the top of the frames and communicating with each filter compartment. Solutions are admitted from a channel in either of the upper corners of the frames. At the middle of the bottom of each frame, and passing through the filter plates, is a continuous channel within which, with ample room to spare, is a supply-pipe for water under pressure. The pipe is provided with nozzles projecting into each compartment. The pipe can be revolved through about 180° so as to play a stream of water upon all portions of the slime-cake in the chamber to be discharged, washing it down into the annular opening formed by the continuous channel through the plates and frames and the water-supply pipe. When the press is being filled and leached, the discharge channel is sealed at both ends." Each press has a plate area of 25 sq. ft. per unit and a capacity of 26 tons.

Presses of this type are in successful operation at the Homestake mine, South Dakota, the Esperanza mill, Mexico, and elsewhere. At the Homestake there are 28 in service, each of which has a capacity of about 70 tons per day.

**The Burt Stationary Filter.**—The Burt stationary filter consists of a steel shell of cylindrical shape and usually 3 ft. 6 in. in diameter by 20 ft. long, sloping at an angle of 45°, which contains a number of circular filter leaves, suspended vertically from the top of the shell. At the lower end of the shell there is an hydraulically-operated door. The leaves are identical in construction with those of a vacuum filter, except as regards their shape. The inside of each leaf is connected by a separate pipe which passes through the shell to a solution header outside.

The pulp is forced into the shell under pressure, and a cake is formed on the leaves. The excess pulp is then displaced by compressed air. The cake is washed by solution under pressure, and the excess solution removed in the same way as the excess pulp. The residue is dislodged by low-pressure air admitted within the leaves. The procedure is, in fact, similar to that used with ordinary leaf filters, except that pressure is applied outside the leaves instead of a vacuum within them, both for forming and washing the cake.

**Burt Revolving Filter.**—In this type the cylinder, which is 3 ft. 6 in. diameter by 40 ft. long, revolves at various speeds. It is lined inside with filter mats, curved to fit the interior. Corrugated boards are placed inside the mats to hold the canvas sides against the pressure. The pulp and wash solution enter through a hollow trunnion, at one end of the cylinder, and the cake is discharged through a door at the other end.

In practice, a suitable quantity of pulp is admitted, and the air is applied until the whole of the slime is caked. This point is indicated by air blowing through. Consequently there is no excess, either of pulp or wash, to be handled, an advantage which is gained by the use of the revolving shell. It is a disadvantage, however, in that the cake is formed on one surface only of each leaf, so that the total available area is only 400 sq. ft. per cylinder. To obtain a reasonable output a thick cake is, therefore, necessary, and

4½ in. is usual. For this reason the filter is more suitable for granular material than for colloidal slime. It has done good work at the plant of the El Oro Mining and Railway Co., Mexico, where the slime averages about 38 per cent. of colloidal and 62 per cent. of granular material.

The following figures were given by V. C. Bennet:<sup>1</sup>

Four revolving filters, 3 ft. 6 in. by 40 ft., filter and wash 385 tons of slime in 24 hours.

|  |              |
|--|--------------|
| Moisture in pulp entering presses . . . . .  | 50 per cent. |
| „ residue discharged . . . . .               | 22 „         |
| Length of cycle—                             |              |
| Charging . . . . .                           | 3.5 min.     |
| Making cake at 45 lb. press . . . . .        | 50 „         |
| Delivering 2 tons of wash solution . . . . . | 2 „          |
| Washing . . . . .                            | 30 „         |
| Discharging. . . . .                         | 4.5 „        |
| Total . . . . .                              | 90 min.      |
| Quantity of slime per cycle . . . . .        | 6 tons.      |
| Water pressure for operating door . . . . .  | 120 lb.      |

The filter cloths are scrubbed once a week with wire brushes, and then washed with a water hose. Once every eight weeks they are treated with dilute HCl. The cloths last, with repatching, from 7 to 8 months.

**Kelly Filter Press.**—This type is similar in principle to the Burt fixed filter. The constructional difference is that the filter leaves are mounted on a movable frame and the cylinder is placed horizontally. The basket of frames is arranged so that it can be removed from the cylinder for the discharge of the residue.

<sup>1</sup> Communicated by Messrs Chalmers and Williams, Chicago.

## CHAPTER XLI.

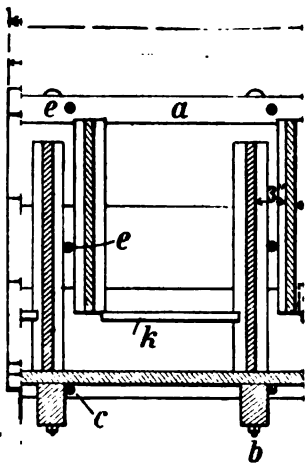
### PRECIPITATION EQUIPMENT.

**Zinc Boxes.**—Data taken from practice for determining the sizes of these, according to the quantity of solution to be dealt with, have been given in a previous chapter, so that it is only necessary here to describe the practical construction of the boxes. The box shown in Figs. 158, 159, 160 is a specimen of the type formerly used on the Witwatersrand. The sides, bottom, and ends are  $1\frac{1}{2}$ -in. material, and this thickness should be used for all smaller sizes as well, because thinner wood is liable to warp, and so cause leakage. Where timber is cheap or solutions rich, 2-in. planks may be used with advantage for the size illustrated. At the top and bottom of each side is a piece *a*, 3 in. square, to stiffen it, and to receive the vertical bolts *b* along the sides and the cross bolts *c* under the bottom. The upright stiffeners *d* are also 3 in. square, and so spaced that the horizontal bolts *e* through the box pass centrally between each pair of partitions. In cutting the notches *f* in these uprights, care should be taken that the part between the horizontal stiffeners *a* is sufficiently short to allow the vertical bolts to draw the side planks tightly together. The same remark applies to the notches at the ends of the  $4\frac{1}{2}$ -in. by 3-in. cross stiffeners *g* under the bottom. All the timber should be machine-dressed all over, and hand-dressed in addition on abutting edges, which are also grooved and tongued. The tongues are separate, except those on the lower edges of the sides and the outside edges of the bottom, where in each case the tongue is formed on the outside edge of the plank itself, so as to weaken the longitudinal stiffeners *a* as little as possible.

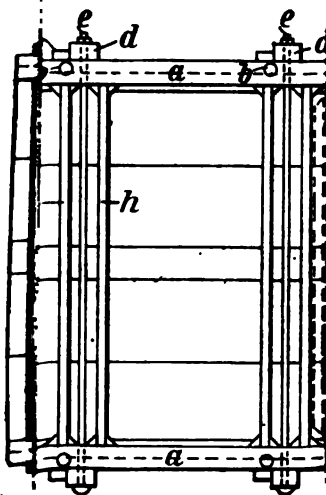
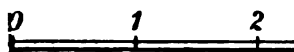
The partitions *h* may be of 1-in. clear pine, the widest obtainable being used so as to have few joints. They must be cut very accurately to the width of the box, in order that they may not prevent the tightening up of the bottom planks, on the one hand, or leave spaces between themselves and the sides, on the other hand. The joints in the partitions are generally tongued and grooved, and the joints between partitions and sides are made watertight by nailing on  $1\frac{1}{2}$ -in. triangular fillets *k*, as shown in plan in Fig. 159.

The box is bolted together with  $\frac{1}{2}$ -in. bolts throughout, but instead of using ordinary  $\frac{1}{2}$ -in. washers, it is better to use plates about  $1\frac{1}{2}$  in. square by  $\frac{3}{8}$  in. thick under the nuts and heads, so as to have a good bearing surface on the timber.

Each compartment has a tray *i*, supported on fillets *k*, nailed to the sides of the box. One such tray is illustrated in Figs. 161, 162. It has sides and



*Part Sec*





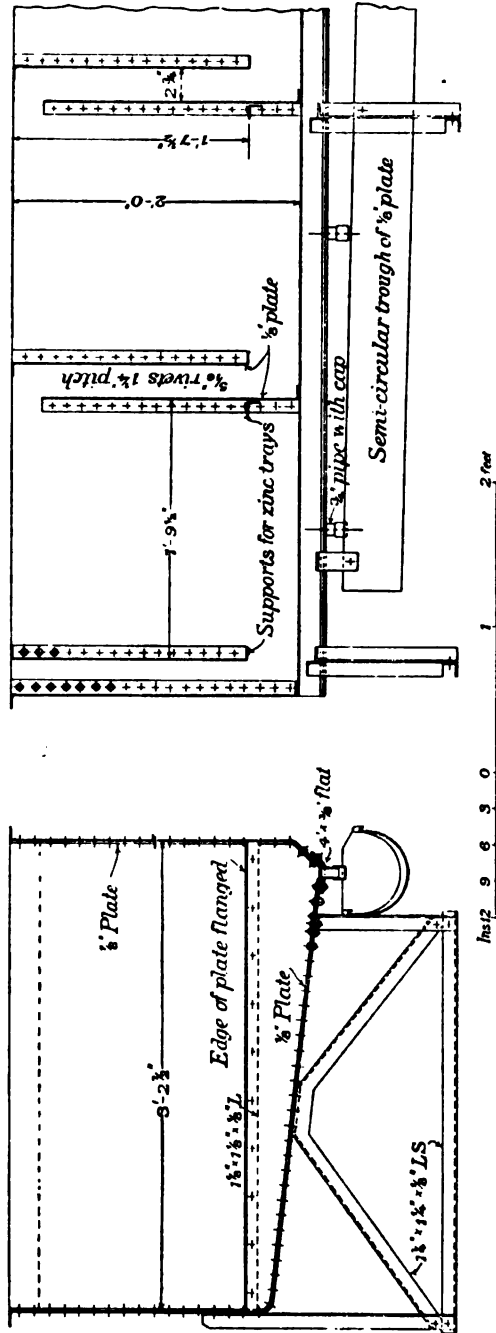
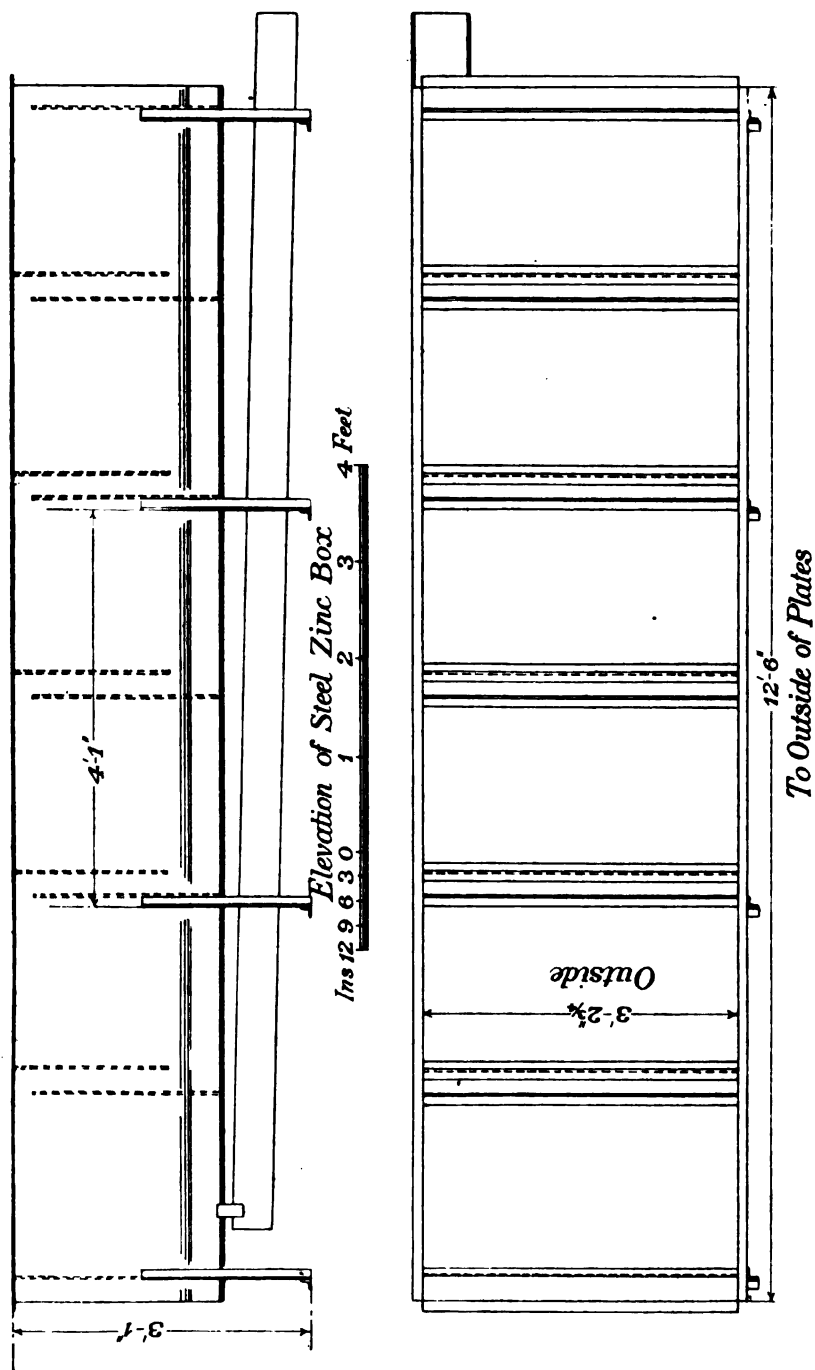


FIG. 163.—Cross-section of Steel Zinc Box.

FIG. 164.—Longitudinal Section of Steel Zinc Box.



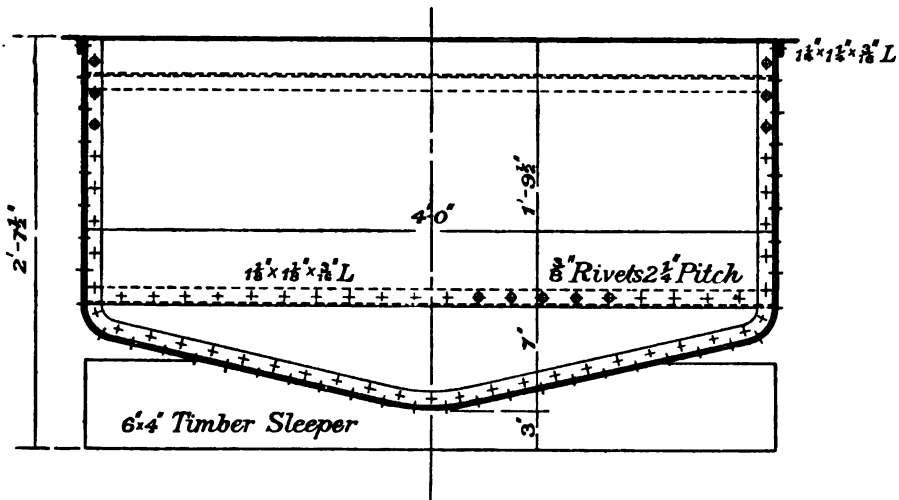


FIGS. 185 and 186.

ends 3 in. deep by 1 in. thick, screwed to triangular corner blocks *l*, and the rectangular frame thus formed is covered below with  $\frac{1}{4}$ -in. screening *m*. Handles *n* of  $\frac{1}{2}$ -in. round iron are attached.

Sometimes zinc boxes are made with a trough along one side. In this case each compartment has a hole in the side fitted with a wooden plug, so that when these plugs are withdrawn the precipitated gold sludge can be washed out of the compartments along the trough into a clean-up vat.

**Steel Zinc Boxes.**—Steel zinc boxes are made with a series of compartments arranged as in the wooden boxes, and are generally built of  $\frac{1}{8}$ - or  $\frac{3}{16}$ -in. plate, according to size. The side plates are flanged outwards at the top to stiffen them, and the end plates and partitions are flanged at each end and riveted to the sides.



**FIG. 167.**—Cross-section of Steel Zinc Box for hand discharging.

The bottom can be inclined at any angle or made trough-shaped, as desired. Steel construction lends itself better to the modern practice of using a few large compartments, instead of many small ones.

Usually, each box has only five or six compartments for the zinc. The width of the box ranges from 3 ft. to 4 ft. 6 in., and the depth of the zinc rarely exceeds 2 ft. The length of each compartment is from 2 to 4 ft., so that the overall length of the box is generally between 12 and 18 ft.

As there is little pressure to withstand, the following riveting is suitable:

|                               |  |                           |
|-------------------------------|--|---------------------------|
| For $\frac{1}{8}$ -in. plate. | $\frac{1}{4}$ -in. rivets.               | $\frac{7}{8}$ -in. pitch. |
| $\frac{1}{8}$ -in. ,,         | $\frac{5}{8}$ - or $\frac{3}{4}$ -in. ,, | $1\frac{1}{4}$ -in. ,,    |
| $\frac{3}{8}$ -in. ,,         | $\frac{3}{4}$ - or $\frac{7}{8}$ -in. ,, | $2\frac{1}{4}$ -in. ,,    |

The dimensions of the box should be designed to suit market sizes, so as to avoid waste.

Figs. 163 to 166 represent a six-compartment zinc box, each compartment of which will contain 8 cu. ft. of zinc. Fig. 167 is a cross section of a steel

box suitable for use when the clean-up is made by hand, in which case it is preferable to have the lowest part in the middle of the compartment. This type is not, however, to be recommended.

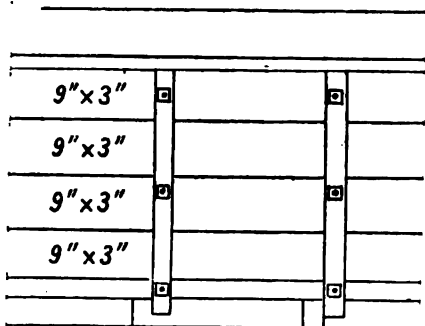
**Electrical Precipitation Boxes.**—The type of box shown in Figs. 168 to 175 was designed for the Siemens and Halske process, but is suitable for use in connection with other types of cathodes and anodes.

The box is 38 ft. long, 6 ft. 1 in. wide, and 3 ft. 1 in. deep, all inside measurements. It is built of 12 in. by 3 in. Oregon pine, with all joints in outer shell grooved and tongued with clear pine tongues  $1\frac{1}{2}$  in. by  $\frac{1}{2}$  in. Each side is finished off on top with a strip *a*,  $4\frac{1}{2}$  in. by 2 in., laid on the flat. Material of the above dimensions is suitable for practically any size of box that is likely to be required.

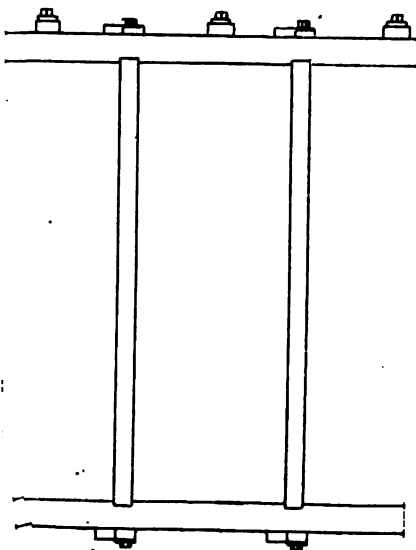
There are fifteen pairs of partitions of 1-in. clear pine, grooved and tongued  $\frac{3}{4}$  in. by  $\frac{1}{4}$  in. These partitions are let into grooves  $\frac{1}{2}$  in. deep in the sides of the box to make a watertight joint, and at the bottom the joints are made good by a triangular fillet. That side of each partition which faces towards the electrode compartments is covered with vertical strips *b b*, 2 ft. 2 in. long by  $3\frac{1}{2}$  in. wide by  $\frac{3}{4}$  in. thick, with  $\frac{1}{2}$ -in. spaces between them and  $\frac{3}{4}$ -in. spaces next to the sides of the box. These spaces form vertical grooves to receive the iron anodes *f f*, which rest on cross pieces of timber *g*, fixed at a suitable height. All the timber is machine-dressed on all faces and edges. The sides are stiffened by upright pieces *c c*, 3 in. square, and the bottom by cross pieces *d d*,  $4\frac{1}{2}$  in. by 3 in. The bolts throughout are of  $\frac{5}{8}$ -in. round iron, screwed at each end, the washers being 2 in. square and  $\frac{1}{4}$  in. thick. There are 17 horizontal cross pieces *e e* of hardwood, 3 in. wide by 2 in. deep, one being fixed across the top of each narrow compartment and one across the foot of the box. Each strip has a groove, 6 ft. long,  $\frac{3}{4}$  in. deep, and  $\frac{3}{8}$  in. wide, cut in the top of it, which serves to hold mercury for making the electrical connection of the electrodes.

The iron anode plates are sewn up in hessian, but a tinned iron wire, No. 5 S.W.G., is previously brazed to one corner of each plate. This wire is so shaped that when the plate is let down the grooves between *b b* to its proper position, the free end of the wire dips into the mercury trough above mentioned. The support for each lead-foil sheet (see Fig. 172) consists of a single horizontal wire *i*, of the same size, which is bent at the ends so that one end rests in a mercury trough on one side of the large compartment, and the other end of the wire drops in a hole in one of the vertical strips *b b* on the other side of the same compartment.

For the purpose of readily dealing with the deposit of sludge, each compartment is provided with a mudhole *h*, fitted in the following manner : (see Fig. 175). A 3-in. hole is bored through the side, so low down that it cuts slightly into the bottom, which is also cut to fit a 3-in. flange, care being taken to place the hole so that the cutting for the flange does not go through the projecting part *j* of the plank. The flange is bolted to the box on the outside of the hole, and fitted with a nipple, socket, and plug. By this

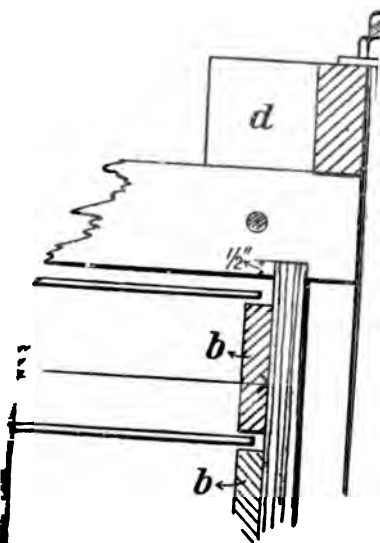


8 9 10 Feet



erated Plan.





precipitation Box.

FIG.



arrangement the outlets project sufficiently from the side of the box, so that a trough can be temporarily placed under them when the sludge is to be withdrawn.

A convenient method of supporting such boxes is to let them rest on three longitudinal bearers,  $4\frac{1}{2} \times 3$  in., laid flat on the cement floor of the shed, with glazed tiles between these bearers and the cross stiffeners  $dd$ , to ensure good insulation.

**Weights of Steel Zinc Boxes.**—The weight of steel in the type of box shown in Figs. 163 to 166 is 1268 lb., or 26·4 lb. per cubic foot of net zinc space. According to design the weight varies in practice from 25 to 40 lb. per cubic foot available for zinc. Makers generally give weight of standard sizes in their catalogues.

**Wooden Boxes.**—For wooden zinc boxes, the quantities may be closely estimated by multiplying the gross contents of the box in cubic feet ( $K$ ) by the coefficients given below.

Let  $C$  = quantity of timber in box in cubic feet.  
 „  $S$  = square feet of timber reduced to 1 in. thick.

Then, on the average,

$$C = K \times 0.35.$$

The variation of the coefficient, for the cases examined, was only from 0.32 to 0.37, and this variation was independent of the sizes.

For square feet 1 in. thick,

$$S = K \times 4.20.$$

For the weight of iron in bolts, nuts, and washers, the coefficient varies. For the zinc box illustrated, the weight in lb. is  $W = K \times 1.13$ ; and for the larger electrical box,  $W = K \times 0.9$ .

**Gold and Silver Precipitate Presses.**—These are used for removing excess moisture from raw zinc-box precipitate or refined acid-treated product. In construction, the plates and frames are similar to those already described in the chapter on pressure filters, although smaller in size. Filling is preferably done through a port at the top of the frames, instead of at the bottom, on account of the higher specific gravity of the material being handled. Washing of the pressed cake is carried out by either of the methods described; and air-drying, until the caked precipitate contains only a few percentages of moisture, is generally practised. In a large number of instances no further drying is practised before smelting operations.

The pumps used for filling clean-up presses are generally attached to the head of the press, and are driven with suitable gearing by a variable-speed engine preferably operated by compressed air. A fairly heavy flywheel is essential on the crank-shaft of the pump so that the slowest speed may be maintained for hardening without unnecessarily raising the pressure. Maximum filling pressure on this type of pump varies from 80 to 120 lb. The size of frame varies with the amount of material to be handled, but the thickness of cake formed seldom exceeds 1 inch.



The chamber press, Fig. 176, which is sometimes used for this purpose, consists of a number of plates *a*, corrugated on each face and provided with projecting flanges *b*, so that when the plates are pressed together, the spaces *c* within the flanges constitute a series of closed chambers for the reception of the material to be filtered. In this type the plates have a central aperture *d*, so that there is a continuous central channel through the press in free communication with each filter chamber. It is necessary to have a

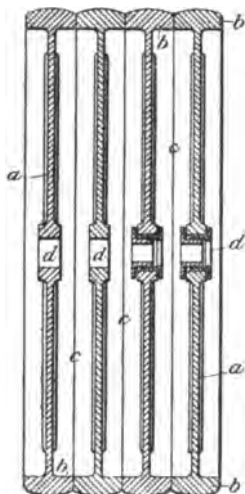


FIG. 176.—Chamber Clean-up Press.

filter cloth on each side of each plate, with a corresponding central hole. In some presses these are fixed in the following manner: Two cloths are sewn together all around these centre holes; one cloth is then pushed through the hole in the plate and opened out, then both are hooked to the top of the plate.

**Merrill Precipitation Presses.**—The usual Merrill precipitation press frame is triangular in shape. At the Homestake<sup>1</sup> there are six of these presses in operation, with sides 48 inches overall and 42 in. vertical height. These presses precipitate from about 2140 tons of solution per 24 hours.

**Zinc- and Aluminium-Dust Feeders.**—A large variety of zinc- and aluminium-dust feeders are in use; and descriptions of apparatus may be found in the technical press,<sup>2</sup> to which the reader is referred.

The earliest mechanical device for feeding zinc dust was a traveling belt, upon which the dust was evenly spread by hand. In some cases this belt was driven by electric motor. In other cases, the motion of the belt was effected and controlled by a float in the gold solution tank. The belt delivered the dust into a small tank or a cone, where, by the use of compressed air, an emulsion was formed with water or solution. The emulsion was delivered to a pipe where it met and mixed with the pregnant solution passing to the precipitation press.

Allan J. Clark<sup>3</sup> found that the use of compressed air for mixing zinc dust with solution results in a heavy deposit of calcium carbonate. He also found that better precipitation results are obtained when the emulsification by air is omitted. He suggests that the calcium carbonate coats the particles of zinc and in that way interferes with the precipitation of the gold and silver.

Later the use of compressed air was abandoned, the solution is deoxidised and the zinc is now being fed by means of various mechanical feeders, a few typical examples of which may be briefly described. These take the place of

<sup>1</sup> "Metallurgy of Homestake Ore," by Clark and Sharwood, *Bull. I. M. and M.*, Nov. 1912.

<sup>2</sup> Allan J. Clark, *Min. Mag.*, April 1911; H. A. and R. T. Sill, *Eng. and Min. Jour.*, April 12, 1913; A. W. Morris, *Min. and Sci. Press*, Feb. 10, 1912; E. M. Hamilton, *Eng. and Min. Jour.*, May 10, 1913; S. F. Kirkpatrick, *Eng. and Min. Jour.*, June 28, 1913.

<sup>3</sup> *Ibid.*

the belt feeder which is irregular in its delivery, especially when the zinc dust is moist.

At the Homestake mine the zinc is placed in a hopper-shaped box, closed at the bottom by the upper surface of a flanged roller. Immediately above this roller there is an opening in the side of the hopper and a sliding gate to regulate the size of the opening and thereby control the feed of zinc dust. The rotation of the roller causes the dust to flow from the hopper in a uniform stream which falls directly into the pregnant solution.

Another feed apparatus described by H. A. and R. T. Sill, includes a cylindrical cone-bottomed storage hopper provided with a revolving feed plate, and a miniature tube mill 6 by 12 in. The barren solution flowing from the precipitation press is measured in a tripping bucket which makes an electrical contact at each trip. The current operates a pawl which drives the feed plate by means of a ratchet wheel. Thus the amount of zinc fed is directly proportional to the rate of flow of the solution. The dust falls from the feeder into the tube mill where it is mixed with solution and cleaned, and whence it passes to the pregnant solution flow.

At the Palmarito mine, Mexico,<sup>1</sup> where the air is moist, the zinc dust is lifted from the storage box by four revolving arms which rotate in a narrow compartment, centrally situated in the storage box and communicating with it on each side by a long slot at the bottom. The arms have bucket-shaped ends to pick up the zinc. The dust in the box is continually stirred, to prevent packing, by other revolving arms in the two outer compartments. In this case also the zinc is passed through a small tube mill.

These devices are, of course, equally applicable for use in connection with aluminium dust.

<sup>1</sup> A. W. Morris, *Min. and Sci. Press*, Feb. 10, 1912.

## CHAPTER XLII.

### SMELTING EQUIPMENT.

**The Cornish Furnace.**—Smelting operations in small properties, and in some cases in large ones, are carried out with an ordinary pot furnace similar to that used for melting bullion or for general assay smelting work. Fuel used is usually either coke or charcoal, or a mixture of both. When charcoal alone is available ample space must be left for the fuel.

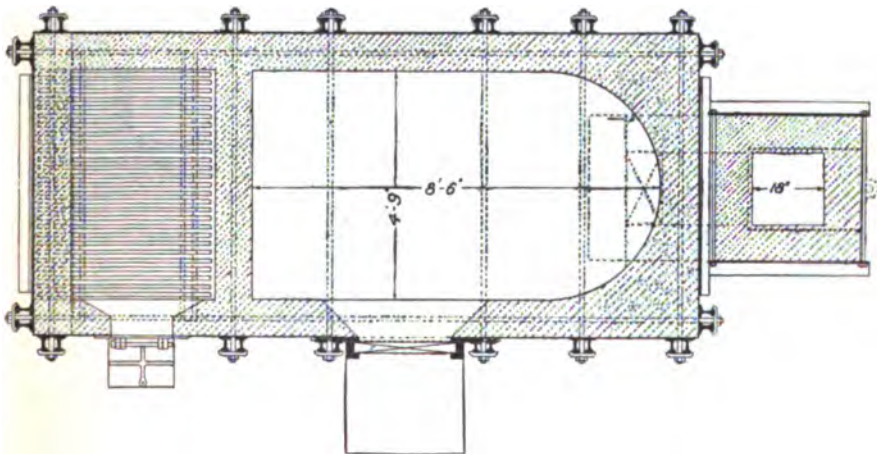
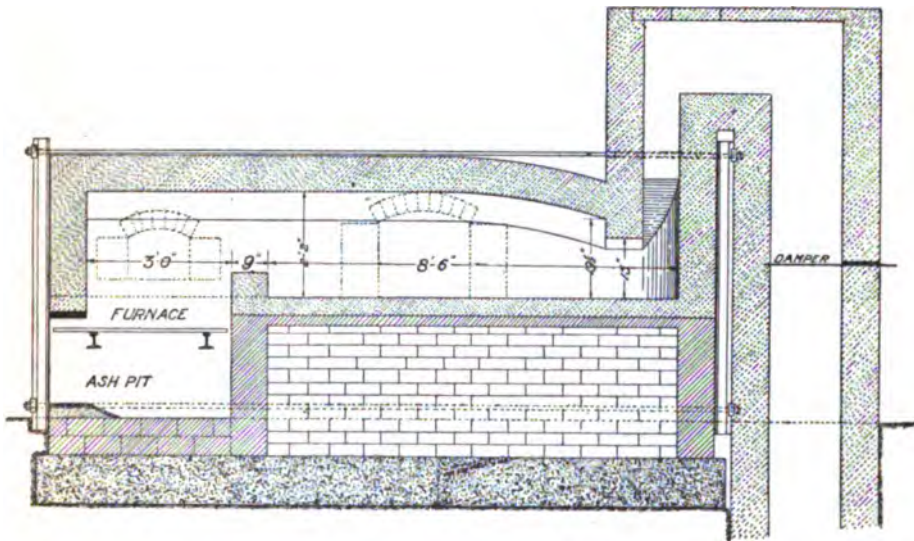
At the Rio Plata mine, in Mexico, the cyanide precipitate is melted in this type of furnace. Large crucibles are used, the charge being regulated so that about 2000 oz. of bullion is poured in one operation. The fuel used is charcoal.

**Reverberatory Furnace.**—This type of furnace, Figs. 177, 178, and 179, taken from *Rand Metallurgical Practice*, Vol. ii., is largely used on the Rand. The openings are on the side and crucibles are passed in and out of the furnace in an upright position during smelting operations. The method is cleanly and has many advantages.

**Lead-Smelting Furnace.**—This type of furnace is used in connection with the Tavener process, and also sometimes features in the smelting treatment of zinc-dust precipitate in the Merrill process. The smelting furnace is built on the same lines as the ordinary reverberatory furnace but with a dished hearth sloping towards a tap hole through which the lead bullion is withdrawn at the conclusion of the smelt.

**Cupelling Furnace.**—In some cases the briquetted precipitate is charged direct on to a bath of molten lead in the cupel furnace; in other cases this furnace is used, as in assaying, for removing the lead from base bullion produced in a reverberatory or blast furnace. A type of cupelling furnace used on the Rand is illustrated in Figs. 180, 181, and 182, which are taken from *Rand Metallurgical Practice*, Vol. ii., to which the reader is referred for details of construction.

**Coke Tilting Furnace.**—This furnace, commonly known as the Faber du Faure, has been largely adopted in West Australia for the smelting of gold precipitate. The furnace consists of a fire-brick chamber, about 4 ft. cube, held together by suitable castings and supported on two bearings placed centrally at the sides. The bottom of the furnace carries fire-bars across the whole width, and is arranged some 2 feet or so above the ground. An arch of firebrick above the bars serves to take the weight of the crucible which rests on this at an angle. The mouth of the crucible, which is bottle-shaped,



FIGS. 177 and 178.—Reverberatory Furnace.

projects through the front wall of the furnace and is bricked in place. The main flue to the stack is permanent, and draught is effected through a contact joint which is broken when the furnace is tilted.

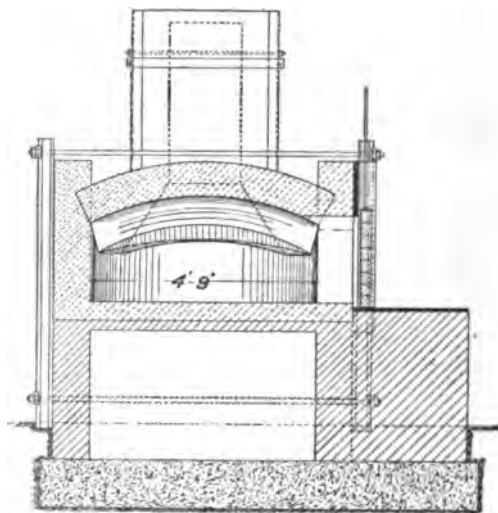
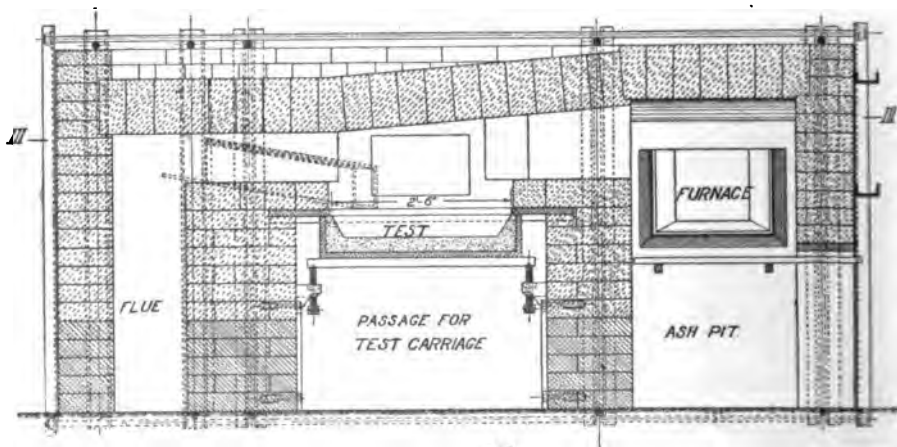


FIG. 179.—Reverberatory Furnace; Cross-section.



SECTION I-I

FIG. 180.—Cupelling Furnace.

The crucibles are generally clay-lined and are capable of taking about 2 cwt. of fluxed precipitate per charge. The furnace is capable of dealing with about  $\frac{1}{2}$  ton of such material per twenty-four hours. A clay-lined crucible, on the average, lasts for about three smelts, each of about twenty-four hours. Firing is done through a door on the top of the furnace which permits of the entrance of large-sized coke, and consequent economy

in fuel consumption. Tilting of the furnace is done by a handwheel which operates through worm gear.

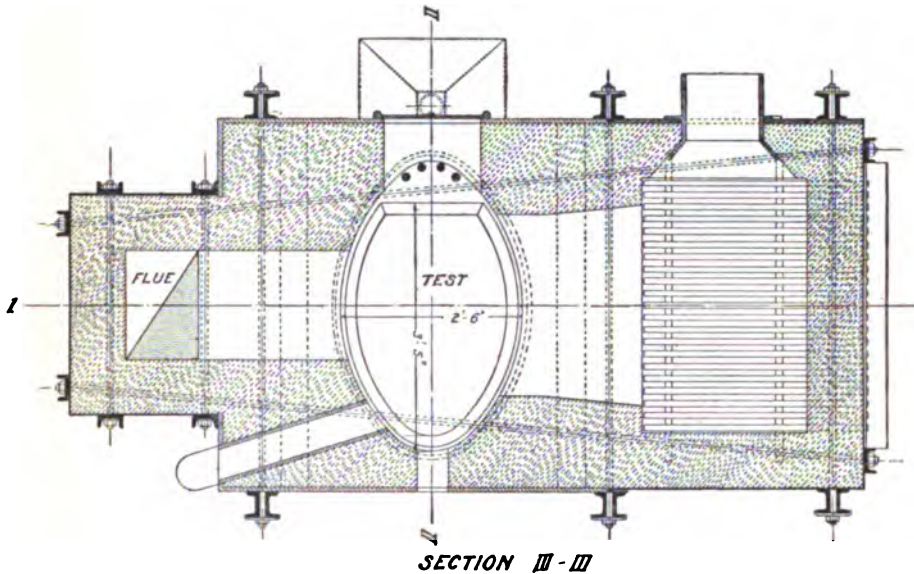


FIG. 181.—Cupelling Furnace ; Plan.

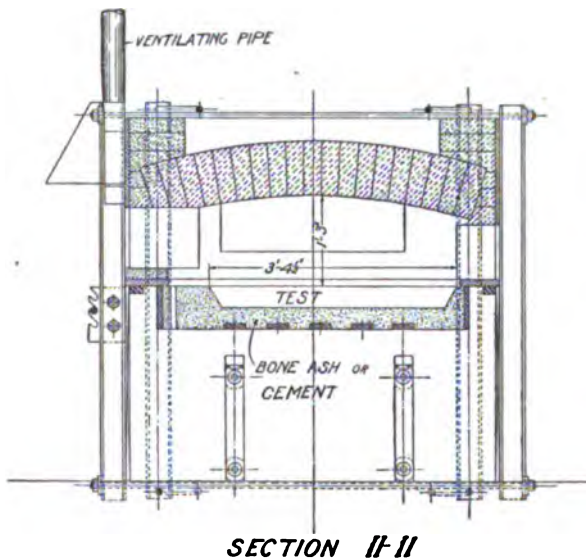


FIG. 182.—Cupelling Furnace ; Cross-section.

**Morgan Constant-Pouring-Point Furnace.**—This furnace, which is adapted for use with coke, oil, or gas as fuel has an additional advantage over similar

types of furnaces in that the fulcrum of tilting is placed close to the lip of the crucible, so that the pouring point remains constant in all positions of tilt.

The oil-fired furnace has been largely adopted of late for the smelting of cyanide precipitates, the advantage of absolute cleanliness of operation being recognised as an important asset in such work. Control of temperature is much more feasible with oil than with other fuel, and this fact offers an additional advantage in the use of such a furnace.

**Electric Furnace.**—H. R. Conklin<sup>1</sup> has used a modified electric furnace at the Lluvio del Oro Mine, Mexico, for the smelting of silver-cyanide precipitate, and has met with considerable success.

<sup>1</sup> *Eng. and Min. Jour.*, June 15, 1912.

## CHAPTER XLIII.

### HANDLING OF MATERIAL AND DISPOSAL OF RESIDUE.

#### *Section I.*

FROM the commercial point of view, this is one of the most important divisions of the subject of cyanide treatment, on account of the large quantities of material that have to be dealt with in most cases. The matter may, for general consideration, be conveniently divided into four separate stages, taken in their natural sequence, namely—

- (a) Bringing material to the works.
- (b) Filling vats.
- (c) Discharging vats.
- (d) Disposal of residues.

Such of the methods and apparatus employed in these four operations as require discussion in detail will be dealt with separately.

(a) In dealing with small heaps, the Australian practice is to use Scotch carts, locally known as drays. These are backed up to the heap, filled by shovelling, then taken to the works and backed, and tipped on to a platform level with the top of the vats. The S. African method of dealing with larger accumulations is, to gradually open out a cutting into the heap for a tram line, and to shovel directly into trucks. When the sand has been piled up to a considerable height, the trucks can be filled so quickly that a single line is generally sufficient. With shallow heaps and in dams, two branch lines are necessary, in order that one set of trucks may be filled while another set is being hauled to the works. In either system, the line is swung around as the material is worked out, so that the trucks are always near the working face. In Fig. 183, two ways of arranging the lines are shown. By method *a*, the loading points gradually become further apart, and the workmen have to walk some distance from one to the other, but at *b* each line serves half the length of the face, and both swing round in the same direction, so that the loading points are always near together, and the work and supervision are more conveniently performed.

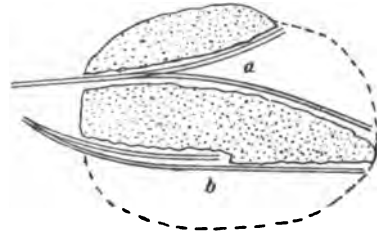


FIG. 183.—Tram Lines.

Sometimes the material is loosened by ploughing, and is brought to the



tram line by means of scrapers drawn by horses in the manner common in railway construction. With slimy material, which requires disintegrating, this is an excellent method, and is also cheap. In Montana, railroad cuttings are made in the ground and covered by bridges with holes over the track, so that the material brought by the scraper is delivered directly through these holes into trucks.

C. W. Merrill says that six men with twelve horses loaded approximately 400 cu. yd. per day, with an average haul of not less than 100 ft. In this case, cars of 3-ton capacity with bottom discharge were used on a track with 56-lb. rails, laid to a 3-ft. gauge. A 22-ton locomotive draws 16 loaded cars up the maximum grade of 3.5 per cent., and the total distance hauled is  $2\frac{1}{2}$  miles. A long line such as this is, however, very unusual in cyanide work, and in most cases side-tipping trucks of about 20 cu. ft. capacity are employed. These are preferably moved by manual labour, animal traction, or ropes worked by a steam winch, when the distances are short and the loading points constantly shifting. When, however, the material can be received and delivered at fixed points, as, for instance, in the transfer from collectors to treatment vats, and the length of the line is considerable, the system of continuous haulage by endless rope is suitable, but only when such large quantities are dealt with that the saving in cost of transport is more than sufficient to pay for maintenance of plant and interest on first cost. This remark applies, of course, to all labour-saving machinery, and it is necessary to insist upon it, because of the tendency so often manifested to introduce such devices for the appearance of economy, even in cases where more primitive methods would be cheaper. And in reference to endless rope haulages, it may be pointed out that even the current costs of transport are not reduced by its introduction when the quantity of material to be shifted is small, because the power absorbed in driving the gearing and rope is out of all proportion to the power consumed in moving the trucks. For short, inclined roads which are too steep for animal traction, a steam winch drawing up trains of trucks is suitable, even for small quantities, because the power is directly employed in doing useful work, and the first cost of the installation is small.

For this purpose, winches from 8 to 12 nominal h.p. are generally used with trains of four or six trucks, each of 20 cu. ft. capacity, on inclines of, say, 1 in 7. If more trucks are used, it means not only a larger winch, but also more men for tipping, or a greater loss of time between the trips. The drums should be from 2 ft. 6 in. to 3 ft. in diameter, and one drum should always be loose on the shaft and provided with a clutch gear. This allows the winch to be used for two loading points at different distances, the working faces being arranged so that the shorter haul is made by the rope on the loose drum. It is also essential that the winch should be large enough for the work, and that it should be well made and properly designed, moreover, with a view to the convenience of the driver—a point often overlooked in small winches.

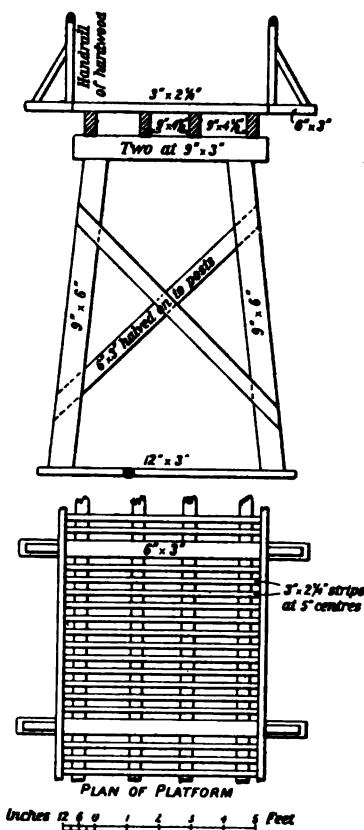
For bringing crushed ore to the treatment vats from extensive heaps, or from dry crushers, and for elevating purposes, belt conveyors are largely used.

When wet material, such as battery pulp, has to be moved, the obvious method where there is sufficient fall is by launders; and when the vats are higher than the source of supply, the lifting appliances generally used are tailing wheels, air lifts, or sand pumps, delivering, as the case may be, to revolving distributors, slat-gate collectors, or classifiers. In Siberia such material is sometimes lifted by the Archimedeian pump—a long cylinder of wood, with a continuous internal thread like a spiral stair. The cylinder is inclined at an angle of about  $40^\circ$ , and the lower end dips into a sump, which is continuously supplied with the pulp. The latter is discharged from the upper end of the cylinder.

(b) The methods of filling pulp into vats have already been sufficiently described. With dry, or nearly dry, material, a good distribution of the material over the whole vat is also desirable, as this tends towards uniformity of texture, and consequently to good leaching. When tipping from trucks, therefore, this should be done from a platform sufficiently high, and so placed as to directly command as much of the vat area as possible, two or more platforms, properly spaced for this purpose, being frequently used. A revolving chute has been employed in connection with belt conveyors in filling vats with dry crushed ore.

**Trestle Staging over Vats.**—Figs. 184 and 185 illustrate a common form of platform used when tipping from trucks into vats. This gridiron platform has the advantage that it can be made wide enough to provide a firm foothold for the workmen when tipping, but at the same time prevents an accumulation of sand upon the gangway, especially when the spaces between the strips are wider underneath than on top. In all parts not directly over the vats, and in all other gangways for workmen, the flooring should consist of 3-in. deals or planks, laid with 1-in. spaces between them, but 1½-in. boards are often used.

**Quantities.**—The amount of timber in such staging may be roughly estimated as follows: For a single track averaging 10 ft. above ground level,



**FIGS. 184, 185.—Trestle Staging for 20-ft. Span.**

allow 3 cu. ft. per foot run of platform. For double track staging, allow 5.75 cu. ft. per foot run. These figures include trestles, bearers, angle bracing, flooring, and hand rails.

**Blaisdell Centrifugal Distributor.**—This machine is designed to aerate the sand as delivered to the vat and to spread it evenly. Its essential feature is the flat disc which is fixed to a vertical spindle and which revolves in a horizontal plane. Angle-iron radiating vanes are riveted on the top of the disc. A cylinder about 15 in. deep, made of thin sheet steel, is fixed as a guard ring around the disc. A belt tripper is mounted on the carriage that supports the distributor, hopper, and driving gear. The sand from the tripper passes through the hopper, and falls on the rapidly rotating disc, from which it is thrown in a spreading shower into the vat.

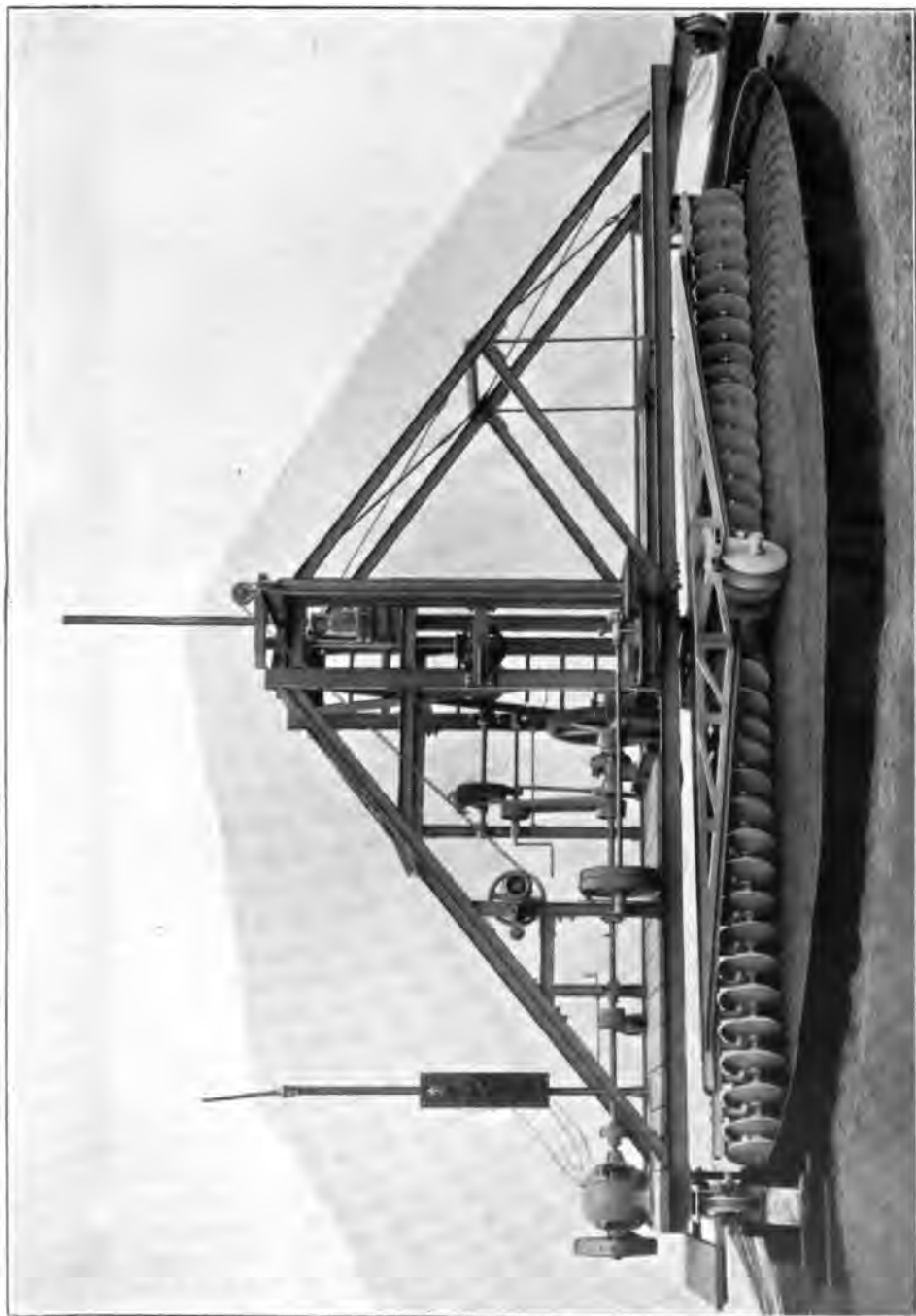
(c) **The Blaisdell Rotary Excavator** is largely used for discharging sand vats. It is shown in Fig. 186. Only one bottom discharge door is required for each vat, situated at the centre. The machine consists essentially of a number of steel rotating discs, similar to those largely used in cultivators, attached to arms, which are rotated from the centre by means of an electric motor. The discs are arranged at an angle with the arm as shown in plan, Fig. 187, and when the arms are rotated in the direction shown by the arrows the discs push the sand inwards to the centre, where it falls down a hole previously made to the belt conveyor beneath. It is obvious that the tendency of the discs is to cause the sand to build up as it approaches the centre, and to counterbalance this action two short arm-carrying discs are arranged at right angles to the long one. This machine is said to have a capacity of 100 tons an hour working in a 30 ft.-vat, and requires 7 horse power.

The Butters distributor, the excavator, and the Blaisdell centrifugal distributor are mounted on carriages that run on a track over the whole length of each series of vats. In order to transfer them to any other series of vats, a transfer table is provided, running on a track at right angles to the main track at the end of the series, as shown in Fig. 187.

**Discharging Sand Residue for Stope Filling.**—This method for the final disposal of treated material is largely practised on the Rand, in Western Australia, and in the United States.

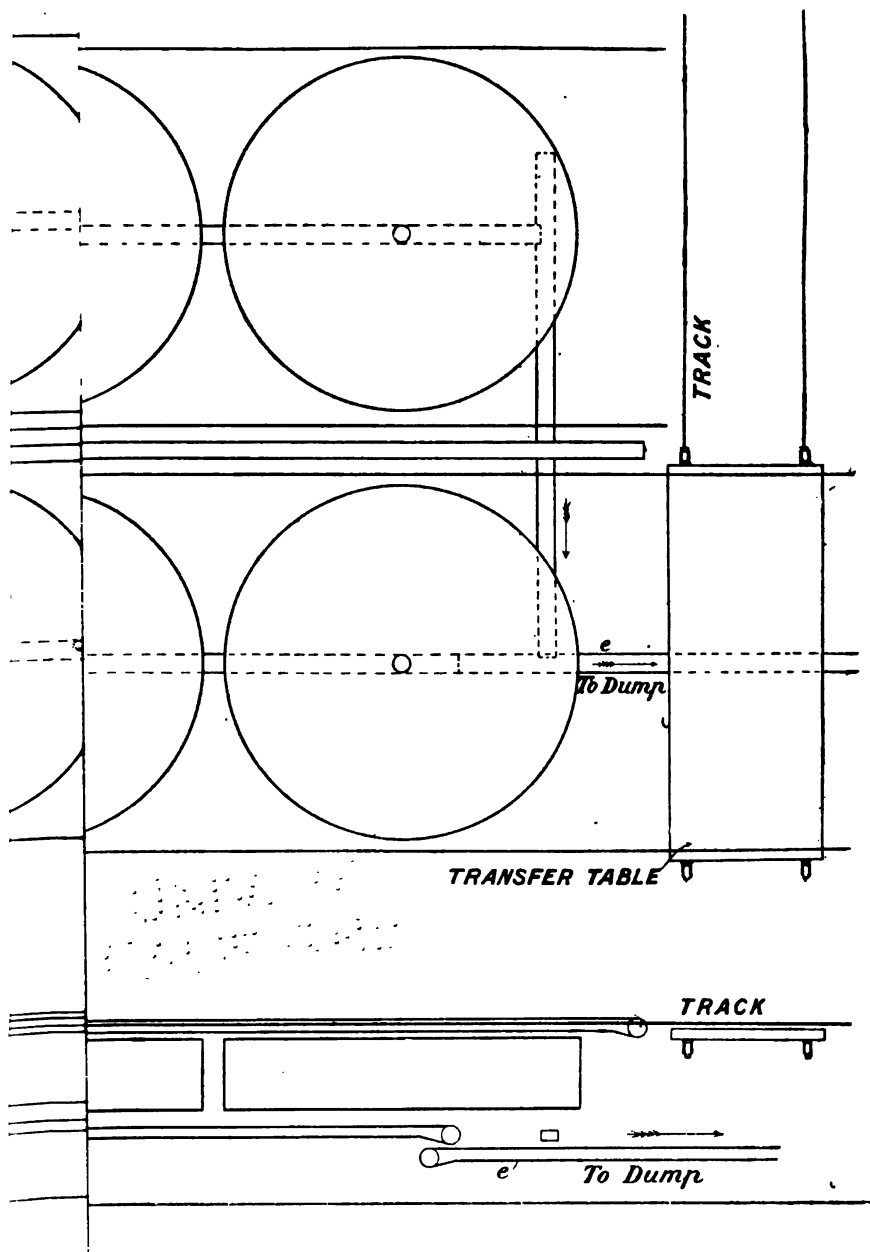
The only point of importance to the metallurgist is the preparation of the sand or slime before delivery to the mine. It is necessary to add a suitable cyanicide, and for this purpose permanganate of potash, in a 5 per cent. solution, is generally used; although in some cases chloride of lime has been adopted for destroying the cyanide.

On the Rand the sand from existing dumps, or preferably direct from the treatment vats, is sprayed with the permanganate solution and sufficient water to give a dilution of about  $3\frac{1}{2}$  to 1. It is then carried by launder or pumped to thickening cones, which discharge a product containing about 30 per cent. moisture. This is delivered from the surface through a bore hole.



**FIG. 186.—The Blaisdell Vat Excavator.**







Edgar Pam<sup>1</sup> says that at the Simmer and Jack mine the addition of 0.02 to 0.025 lb. of permanganate per ton of sand reduced the total cyanide to less than 0.0025 per cent., and that tests of air, from the stopes being filled, showed no trace of HCN.

In Western Australia the residues are filled when dry. A Western Australian Royal Commission recommended that tailing should not contain more than 0.01 per cent. total cyanide, calculated as KCN.

**Trucks.**—In a few cases a 2-ft. gauge has been used in cyanide plants, but experience has shown that the much more common 18-in. gauge is wide enough for all purposes except locomotive work. For small plants a truck

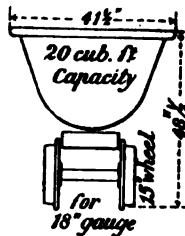


FIG. 188.—Truck.

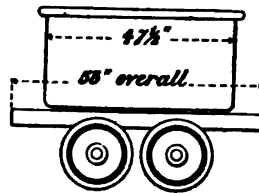


FIG. 189.—Truck.

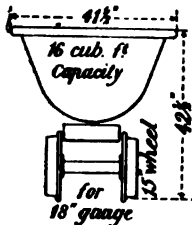


FIG. 190.—Truck.

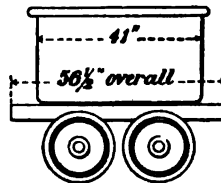


FIG. 191.—Truck.

holding 16 cu. ft. is the usual size, whereas for large plants 20 cu. ft. capacity is generally adopted. When designing works it is often necessary to know the size and weight of the truck which is to be used. The overall dimensions for 18-in. gauge side-tipping V-shaped wagons are given in Figs. 188, 189, 190, 191, and the corresponding weights are as below.

|   |              |
|---|--------------|
| 20 cu. ft. capacity with 15-in. wheels. | Each 860 lb. |
| 16 " " " 15-in. "                       | " 790 "      |
| 16 " " " 12-in. "                       | " 700 "      |

The use of 12-in. instead of 15-in. wheels for the smaller truck reduces the overall height to 39 1/4 inches.

These weights are for trucks built of  $\frac{3}{16}$ -in. plate throughout, and the usual 4 x 2 in. channel frames. Generally speaking, the weight of trucks from 15 to 60 cu. ft. capacity varies from 33 to 50 per cent. of the load

<sup>1</sup> *Jour. C.M. and M. Soc. of S. Africa*, Aug. 10, p. 97.



carried. Those used in cyanide plants usually weigh about 45 per cent. of the net load.

**Tram Lines.**—For temporary tracks that have to be frequently shifted, steel sleepers are undoubtedly the best, but for permanent lines, wooden sleepers 6 in. by  $4\frac{1}{2}$  in. in section and about 3 ft. long give a better hold on the ground. Also, on curves of short radius, wooden sleepers should be used, because they easily allow of the slight increase of gauge which is desirable, but with steel sleepers this alteration involves some trouble and expense. Of course, sharp curves must be avoided as far as possible, as they cause enormous wear of axles, bearings, and wheel flanges. The minimum curve that can be used in practice with 18-in. gauge is about 12 ft. radius, but

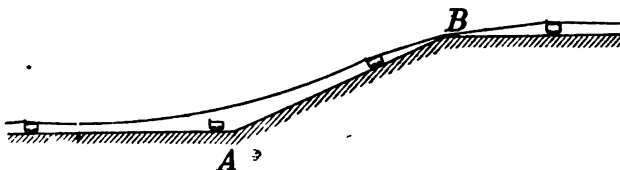
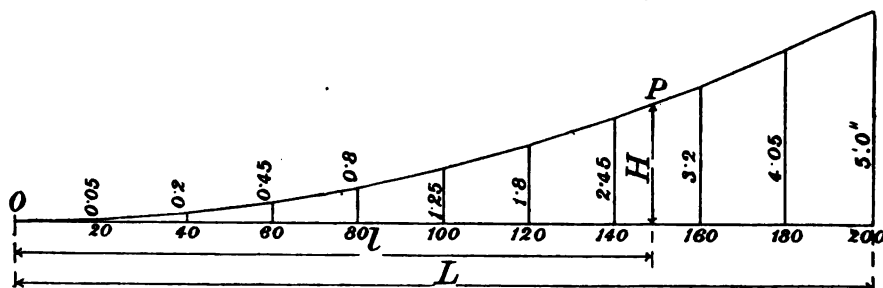


FIG. 192.—Defect in Grading for Rope Haulage.



Vertical Scale = 10 times Horizontal Scale.

FIG. 193.—Method of Grading for Rope Haulage.

this is inadvisable unless space is absolutely limited, and a minimum of 30-ft. radius should be required in laying out permanent lines.

As regards size of rail, 12 lb. per yard is the lightest that should be used, and is heavy enough for shifting lines on dumps and the like; but 14 or 16 lb. is suitable for permanent work. The width of the bottom flange ought to be about the same as the depth of the rail. Light rails are marketed in which depth is much greater than width, but these should be carefully avoided, because, in light rails, lateral stiffness is quite as important as vertical carrying power. This may seem to be a minor detail, but it makes considerable difference in practice. In laying these lines, the track should be well stretched; that is to say, after every two or three lengths have been connected, they should be forced apart as far as the holes in rails and fish-plates will allow.

In laying out tracks, the following points should be kept in mind. For animal traction, uniformity of grade is of great importance, and in order to

combine this with cheap construction, the best method is to lay out on the natural surface of the ground, the curved line which is found by trial with the levelling staff to correspond to the grade selected.

For continuous-rope haulage, the chief point is to make the line as straight as possible, moderate changes of gradient being relatively much less important. If the track cannot be laid in one straight line, then it should be made up in long straight lengths, with short definite curves at the places where the direction changes.

Abrupt changes of grade, such as those shown in Fig. 192, should, however, be avoided. The hollow at A is bad, because the tension in the rope prevents it from following the profile of the track, and it tends to lift out of the grips as the truck approaches the point A. A sudden change of grade of the kind shown at B causes a heavy pressure on the track rollers at that point, with consequent wear of both rope and rollers. In each case a series of changes of grade of less amount should be used, or a curved profile substituted.

At B the shape of the curve is not important, but at A a parabola, as shown in Fig. 193, is suitable, and it can be set out with sufficient accuracy by means of the following simple formulæ :

In the figure O is the point where the change of grade commences, the track to the left of O being level. H is the vertical height of any point P above O, and  $l$  is the horizontal distance to the same point, both in feet.

Let  $T$  = maximum tension in lb. acting in the rope at O, and  
 $w$  = weight of rope in lb. per foot.  
 $g$  = grade of track at any point P expressed as a fraction, and  
 $G$  = final grade to be attained at the upper end of the curve.  
 $L$  = total horizontal length of the curved portion.

$$\text{Then } H = \frac{wl^2}{2T} \quad . \quad . \quad . \quad . \quad . \quad . \quad (1)$$

$$g = \frac{wl}{T} \quad . \quad . \quad . \quad (2) \quad G = \frac{wL}{T} \quad . \quad . \quad . \quad . \quad (2a)$$

$$L = \frac{GT}{w} \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

$$\text{and } l = \sqrt{\frac{2HT}{w}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

As an example, suppose the change to be from a level line to a grade of 5 per cent., with a rope weighing  $1\frac{1}{2}$  lb. per foot and a maximum tension at O of 6000 lb.

Then by (3)

$$L = \frac{1 \times 6000}{20 \times 1.5} = 200 \text{ ft.}$$

The height H at 20 ft. from O by (1) is

$$\frac{1.5 \times 20 \times 20}{12,000} = 0.5 \text{ ft.}$$

and at 40 ft.

$$\frac{1.5 \times 40 \times 40}{12,000} = 0.2 \text{ t.}$$



**Equalising Tractive Force.**—When a residue dump is to be formed at a lower level than the treatment vats, and, generally, when material has to be moved downhill by animal traction, it is advantageous to lay out the grades so that the pull for loaded trucks down the incline is equal to the pull required for the empties up the incline. When a single line is used, and whenever the outgoing and return lines are of the same length, the grade necessary for equal tractive force in either direction can be found as follows :

$W$  = weight of loaded trucks, and  $w$  = weight of empties.

Then, pull down the incline =  $W(K - G)$ , and pull up for empties =  $w(K + G)$ .

Equating these, and transposing, then

$$G = \frac{K(W - w)}{W + w} \quad (9)$$

Taking the weight of a 20-ft. truck at 800 lb., and its contents as 105 lb. per cubic foot, including moisture, the required grade is 1 in 88.

In some cases the return line is shorter than the down line, and must therefore have a steeper gradient in order to reach the starting-point of the down line at the proper level. When the down grade is  $n$  times the length of the up grade, and  $G$  is the grade of the down line, then the returning gradient must be  $nG$ , and the equation for equal tractive force on both is—

$$G = \frac{K(W - w)}{W + nw} \quad (10)$$

To facilitate further the selection of suitable grades, Table LXVIII. has been calculated from the above equations for the 16- and 20-ft. trucks, whose weights have been already given.

TABLE LXVIII.—*Showing Grades required for equal tractive effort on outgoing and return lines, when loaded trucks run on the down gradient.*

| Weight of Truck in lb. | Cubic feet carried. | Fractions given below show down grade = $G$ when $K = \frac{1}{10}$ . |        |        |        |        |                                 |        |        |        |        |
|------------------------|---------------------|---|--------|--------|--------|--------|---------------------------------|--------|--------|--------|--------|
|                        |                     | Load at 90 lb. per cubic foot.  |        |        |        |        | Load at 110 lb. per cubic foot. |        |        |        |        |
|                        |                     | $n=1.$  | $n=2.$ | $n=3.$ | $n=4.$ | $n=5.$ | $n=1.$                          | $n=2.$ | $n=3.$ | $n=4.$ | $n=5.$ |
| 800                    | 20                  | 1/94  | 1/117  | 1/139  | 1/161  | 1/183  | 1/86                            | 1/105  | 1/123  | 1/141  | 1/159  |
| 720                    | 16                  | 1/100   | 1/125  | 1/150  | 1/175  | 1/200  | 1/91                            | 1/111  | 1/132  | 1/152  | 1/173  |
| 610                    | 16                  | 1/92  | 1/114  | 1/135  | 1/156  | 1/177  | 1/85                            | 1/102  | 1/119  | 1/137  | 1/154  |

On short, circular tracks, such as those from collecting vats to treatment vats, and from treatment vats to residue dump, it is better to make the grade steeper than is given above when manual labour is employed, because

men work better intermittently. In fact, men prefer to work on a grade at which the loaded trucks will run down by their own weight, although this necessarily involves harder work in pushing the empty trucks back again. A gradient of 1 in 50 or 60 is therefore suitable for the full trucks in such cases.

**Engine Planes. Power.**—These may be worked at a speed of six to ten miles an hour. In cases where an extra steep grade occurs on a small portion of the whole length, a speed of four miles may be allowed for, on this portion only, when calculating the power which has to be provided. Of course, where small quantities are handled in this way, slower speeds may be adopted in order to economise in the first cost of the hauling winch.

The horse power required in any given case is found by multiplying the tractive force in pounds for the steepest gradient on the line, by the speed in feet per minute on that gradient, and dividing by 33,000.

$$\begin{aligned} \text{Thus if } W &= \text{total weight in lb., including trucks and rope,} \\ G &= \text{as before, grade expressed as a fraction,} \\ M &= \text{speed in miles per hour, and} \\ P &= \text{actual h.p. to be provided, then} \\ P &= 0.0027MW(K+G) \quad . \quad . \quad . \quad . \quad . \quad (11) \end{aligned}$$

It is to be noted, however, that while the maximum grade is used to determine the necessary power of the winch, only the average grade is to be considered when calculating the mean power expended during the whole time of hauling.

So far, only a single set of trucks drawn *up* the incline has been considered, but when a set of empties is lowered at the same time, this will lessen the work of the winch. Let  $p$  = power derived from fall of empty trucks whose weight is  $w$ . Then

$$p = 0.0027Mw(G-K) \quad . \quad . \quad . \quad . \quad . \quad (12)$$

and the total nett h.p. is therefore

$$P - p = 0.0027M\{W(K+G) - w(G-K)\} \quad . \quad . \quad . \quad . \quad . \quad (13)$$

In most cases, however, it is better, when deciding upon the power to be provided, to neglect altogether any gain from this source, and to make provision for the maximum possible load by using equation (11).

On flat inclines, where a tail rope has to be used, equation (13) is applicable also, and should be employed for double lines with two sets of trucks, because in such cases the empty trucks add to the power required from the engine.

In any case, at least 10 per cent. extra power must be allowed for engine friction, starting the trucks, and other matters which do not admit of close calculation. When the winch is situated at a distance from its boiler, the consequent loss of pressure in the steam pipes must also be taken into account.

**Power for Endless-Rope Haulages.**—Equation (13) is also useful in calculating the power required for continuous-rope haulage when the weights

upon the up and down grades respectively can be accurately ascertained, but it only applies strictly to the trucks and their contents.

However, in this case the friction of the rope, driving gear, and road rollers becomes an important item, which may often account for more than half the total power. Some engineers recommend the estimation of this by taking the total weight of all ropes, pulleys, and other moving parts, and calculating in the same way as for the trucks, but with a coefficient of  $\frac{1}{16}$  instead of  $\frac{1}{80}$ . The only safe course is to have plenty of spare engine power, which, after all, only adds slightly to the first cost of the whole installation, and it also provides for any temporary abnormal loading, due to variations in the supply of trucks to the line.

## CHAPTER XLIV.

### HANDLING OF MATERIAL AND DISPOSAL OF RESIDUE.

#### *Section II.*

Of the many systems of rope haulage which are used for different purposes, only two are usually of service in connection with cyanide works, namely, inclined roads worked by steam winches and continuous-rope haulage, with the rope usually *over* the trucks. Consequently only those points in connection with ropes which are of importance in the use of these two systems will be dealt with.

Steel wire ropes are usually made of either—

1. Bessemer steel, with a breaking stress of about 35 tons per square inch of the section of the rope.
2. Siemens-Martin steel, with a breaking stress of about 50 tons per square inch ; and
3. Crucible case steel, with a breaking stress of 90 tons per square inch.

The last is the best for the purpose in view. It is evidently of importance, in ordering rope, either to specify the material of which it is to be made, or to have a guarantee as to its strength.

It will be seen later, in considering the stresses in the rope caused by bending it, that in most cases it is desirable to have a rope made of small wires and of small sectional area. The principal reason why wire of great tensile strength should be used, is that the rope and its constituents may be as small as is possible consistent with the necessary provision for the wearing down of the outer wires by friction against the drums and sheaves.

Another reason for employing wire of great tensile resistance is that a rope of given strength is lighter than a rope of equal strength made of inferior material.

For haulage purposes, ropes are usually made of six strands round a central core of hemp, and each strand contains either 7, 12, 19, or 24 wires.

Table LXIX. gives the breaking weight of such steel wire ropes, for all the sizes likely to be required for the purpose, calculated at 90 tons per square inch. The breaking weights are given in pounds, so that they may be readily converted either into long or short tons.

**Working Stress on Rope.**—This consists of two principal items, first the tractive pull found in the manner already described, and, secondly, the stress due to the bending of the rope round the winding drum or the sheaves and pulleys on the track. This second item is an important one, but is too often

neglected in practice, or rather it is allowed for in a rough and ready way, either by taking a large factor of safety, or by some empirical rule as to the ratio between the diameter of the rope and the minimum diameter of sheave which may be used. But as this bending stress is capable of fairly exact calculation, it is far better to ascertain it, and thereby to know more accurately the actual total stress which is being put upon the rope in any given case.

TABLE LXIX.—*Showing calculated Breaking Weights, in lb., of Steel Wire Ropes at 90 long tons per square inch.*

| Standard Wire Gauge number. | Diameter of Wire in decimals of an inch. | 6 Strands, each of 7 Wires. | 6 Strands, each of 12 Wires. | 6 Strands, each of 19 Wires. | 6 Strands, each of 24 Wires. |
|-----------------------------|--|-----------------------------|------------------------------|------------------------------|------------------------------|
| 25                          | ·020                                     | ...                         | ...                          | ...                          | 9,529                        |
| 24                          | ·022                                     | ...                         | ...                          | 8,785                        | 11,084                       |
| 23                          | ·024                                     | ...                         | 6,567                        | 10,397                       | 13,183                       |
| 22                          | ·028                                     | ...                         | 8,939                        | 14,153                       | 17,868                       |
| 21                          | ·032                                     | 6,809                       | 11,678                       | 18,482                       | 23,846                       |
| 20                          | ·036                                     | 8,619                       | 14,775                       | 23,394                       | 29,550                       |
| 19                          | ·040                                     | 10,640                      | 18,240                       | 28,880                       | ...                          |
| 18                          | ·048                                     | 15,322                      | 26,267                       | 41,589                       | ...                          |
| 17                          | ·056                                     | 20,855                      | 35,751                       | ...                          | ...                          |
| 16                          | ·064                                     | 27,239                      | 46,695                       | ...                          | ...                          |
| 15                          | ·072                                     | 34,474                      | ...                          | ...                          | ...                          |
| 14                          | ·080                                     | 42,560                      | ...                          | ...                          | ...                          |

**Stress due to Bending.**—When a wire rope is bent over a pulley, as shown in Fig. 194, it is clear that although the centre line  $ab$  of the rope retains its original length, the wires on the outer side  $cd$  must be lengthened, and those on the inner side  $ef$  must be shortened. That is to say, there is a tendency to set up a tensile stress in the outer wires and a compressive stress in the inner wires. But the wires of each strand are twisted round the strand, and the strands are twisted round the rope so that each wire occupies alternately the inner and outer position. Consequently the tendency to extension and compression occur alternately in the same wire and neutralise each other, thus, as Weisbach has pointed out,<sup>1</sup> increasing the probability that the tension is uniform in all the wires. But, although there is probably no stress due to the bending of the rope considered as a whole, there is a stress set up in each wire due to the bending of the wire itself. Thus, if we now consider Fig. 194 to represent the bending of a single wire, it is evident that the outer fibre  $cd$  will be in tension and the inner fibre in compression. The former will be added to the tension already existing in the wire due to its share of the tractive force, and the sum of these two tensions will represent the maximum



FIG. 194 — Stresses on Ropes.

<sup>1</sup> *Mechanics of Engineering*, vol. iii. part i., 1890, p. 556.



stress on the wire. It is true that this maximum stress is only exerted on the outer fibres of each wire, but it is also true that if these outer fibres are overloaded, the whole wire will be destroyed.

The stress  $S$  in a wire of diameter  $d$  produced by bending in the arc of a circle of diameter  $D$  is given by the expression

$$S = \frac{Ed}{D},$$

where  $E$  is the modulus of elasticity of the steel.<sup>1</sup> The two diameters  $d$  and  $D$  must of course be expressed in the same units, such as feet or inches, and the stress will be given in the same units as the modulus of elasticity.

This formula brings out two important points very clearly, viz.—

1. That the stress in the wires caused by bending is inversely proportional to the diameter of the sheave round which the rope is bent, and therefore the largest sheaves that are practically convenient should be used.

2. That the stress is directly proportional to the diameter of the wire of which the rope is composed, and consequently, if for any reason it is necessary to use small sheaves, a rope composed of comparatively small wires is to be preferred in such a case.

To apply the above formula in practice for any rope it is necessary to find the stress ( $S$ ) per square inch, and then multiply the figure thus found by the total sectional area of all the wires of the rope, expressed in square inches.

One example will make this quite clear, and the case of a rope which consists of 6 strands and 7 wires may be taken, each wire being of No. 16 S.W.G., that is, 0.064 in. in diameter. Assuming a 3-ft. pulley and taking  $E = 29,000,000$ , the above formula becomes

$$S = \frac{29,000,000 \times 0.064}{36} = 51555 \text{ lb. per square inch.}$$

The sectional area of each wire is 0.003217 sq. in., so the total section of the 42 wires = 0.135114.

$$\text{And } 51555 \times 0.135114 = 6914 \text{ lb.}$$

In the Tables LXX. to LXXII. these stresses will be found for all the sizes of rope included in Table LXIX., and for the sizes of pulleys which are likely to be used in practice.

In view of the important relation that exists between the bending stresses and the size of the individual wires, it is unfortunate that makers of wire ropes, as a rule, give no information as to these sizes in their catalogues, but are content to state only the circumferences of the whole ropes, which, of course, vary with each particular make. For instance, one maker catalogues a rope of 42/19 as  $1\frac{3}{4}$  in. in circumference, and another maker has a rope of 42/20 described as  $1\frac{3}{4}$  in.

<sup>1</sup> For proof of this formula in the form, tension =  $\frac{Ed}{2R}$ , see *Transmission of Power by Wire Ropes*, by Albert W. Stahl.

TABLE LXX.—*Bending Stresses in Wire Ropes composed of Six Strands, with Seven Wires in each Strand.*

| Diameter of Sheave. | 21<br>S.W.G. | 20   | 19    | 18    | 17    | 16    | 15    | 14    | No. S.W.G.  |
|---------------------|--------------|------|-------|-------|-------|-------|-------|-------|---|
|                     | •032         | •036 | •040  | •048  | •056  | •064  | •072  | •080  | Diam. of Wire, in.  |
| 6 ins.              | 5224         | 7439 | 10208 | 17632 | ...   | ...   | ...   | ...   | These numbers represent total stress in lb. due to bending. |
| 1 ft. 0 "           | 2612         | 3719 | 5104  | 8816  | 14000 | 20896 | ...   | ...   |   |
| 1 " 6 "             | 1741         | 2480 | 3408  | 5874  | 9333  | 13932 | 22401 | 27210 |   |
| 2 " 0 "             | 1306         | 1860 | 2552  | 4408  | 7000  | 10449 | 16801 | 20408 |   |
| 2 " 6 "             | 1045         | 1488 | 2042  | 3526  | 5600  | 8359  | 13441 | 16326 |   |
| 3 " 0 "             | 871          | 1240 | 1701  | 2939  | 4667  | 6914  | 11201 | 13605 |   |
| 3 " 6 "             | ...          | 1063 | 1458  | 2519  | 4000  | 5971  | 9601  | 11661 |   |
| 4 " 0 "             | ...          | 930  | 1276  | 2204  | 3500  | 5244  | 8401  | 10204 |   |
| 4 " 6 "             | ...          | ...  | 1184  | 1948  | 3111  | 4644  | 7467  | 9070  |   |
| 5 " 0 "             | ...          | ...  | 1021  | 1763  | 2800  | 4180  | 6720  | 8163  |   |
| 5 " 6 "             | ...          | ...  | ...   | ...   | 2515  | 3800  | 6109  | 7421  |   |
| 6 " 0 "             | ...          | ...  | ...   | ...   | ...   | ...   | 5600  | 6803  |   |

TABLE LXXI.—*Bending Stresses in Wire Ropes composed of Six Strands, with Twelve Wires in each Strand.*

| Diameter of Sheave. | 23   | 22   | 21   | 20    | 19    | 18    | 17    | 16    | No. S.W.G.  |
|---------------------|------|------|------|-------|-------|-------|-------|-------|---|
|                     | •024 | •028 | •032 | •036  | •040  | •048  | •056  | •064  | Diam. of Wire, in.  |
| 6 ins.              | 3778 | 6000 | 8956 | 12752 | 17492 | ...   | ...   | ...   | These numbers represent total stress in lb. due to bending. |
| 1 ft. 0 "           | 1839 | 3000 | 4478 | 6376  | 8746  | 15099 | 23999 | ...   |   |
| 1 " 6 "             | 1259 | 2000 | 2985 | 4251  | 5831  | 10066 | 16000 | 23833 |   |
| 2 " 0 "             | 945  | 1500 | 2239 | 3188  | 4373  | 7550  | 12000 | 18912 |   |
| 2 " 6 "             | 756  | 1200 | 1791 | 2550  | 3948  | 6040  | 9600  | 14330 |   |
| 3 " 0 "             | 630  | 1000 | 1493 | 2125  | 2915  | 5033  | 8000  | 11942 |   |
| 3 " 6 "             | ...  | ...  | 1279 | 1822  | 2499  | 4314  | 6857  | 10236 |   |
| 4 " 0 "             | ...  | ...  | 1119 | 1594  | 2187  | 3775  | 6000  | 8956  |   |
| 4 " 6 "             | ...  | ...  | ...  | ...   | 1944  | 3355  | 5338  | 7961  |   |
| 5 " 0 "             | ...  | ...  | ...  | ...   | 1749  | 3020  | 4800  | 7165  |   |
| 5 " 6 "             | ...  | ...  | ...  | ...   | ...   | 2745  | 4364  | 6514  |   |
| 6 " 0 "             | ...  | ...  | ...  | ...   | ...   | 2517  | 4000  | 5971  |   |

*Note.*—For ropes composed of six strands of twenty-four wires in each strand, the stresses will be just double the figures given in this table.

TABLE LXXII.—*Bending Stresses in Wire Ropes composed of Six Strands, with Nineteen Wires in each Strand.*

| Diameter of Sheave. | 24   | 23   | 22   | 21    | 20    | 19    | 18    | No. S.W.G.  |
|---------------------|------|------|------|-------|-------|-------|-------|---|
|                     | •022 | •024 | •028 | •032  | •036  | •040  | •048  | Diam. of Wire, in.  |
| 6 ins.              | 4608 | 5983 | 9501 | 14180 | ...   | ...   | ...   | These numbers represent total stress in lb. due to bending. |
| 1 ft. 0 ins.        | 2304 | 2991 | 4750 | 7090  | 10096 | 13848 | 23930 |   |
| 1 " 6 "             | 1536 | 1994 | 3167 | 4727  | 6730  | 9232  | 15953 |   |
| 2 " 0 "             | 1152 | 1496 | 2375 | 3545  | 5048  | 6924  | 11965 |   |
| 2 " 6 "             | 922  | 1197 | 1900 | 2836  | 4038  | 5539  | 9572  |   |
| 3 " 0 "             | 768  | 997  | 1583 | 2363  | 3365  | 4616  | 7977  |   |
| 3 " 6 "             | ...  | 855  | 1357 | 2026  | 2884  | 3956  | 6837  |   |
| 4 " 0 "             | ...  | 748  | 1178 | 1773  | 2524  | 3462  | 5932  |   |
| 4 " 6 "             | ...  | ...  | ...  | 1576  | 2243  | 3071  | 5322  |   |
| 5 " 0 "             | ...  | ...  | ...  | 1418  | 2019  | 2770  | 4785  |   |
| 5 " 6 "             | ...  | ...  | ...  | ...   | ...   | 2518  | 4351  |   |
| 6 " 0 "             | ...  | ...  | ...  | ...   | ...   | 2308  | 3999  |   |

**Factor of Safety.**—When the maximum working load on the rope and the stresses due to bending have been carefully ascertained, it is not necessary to use such large factors of safety as are required when the loads and stresses are only roughly estimated. In the majority of cases in connection with metallurgical works a factor of safety of 4 is amply sufficient for the working load. In cases of inclines, where any danger to workmen would result from the breaking of the rope, a factor of safety of 5 or 6 should be allowed. In all cases a factor of 3 is sufficient to allow on the calculated bending stress.

### *Examples.*

A few examples will illustrate the most convenient method of using the above tables.

*Example 1a.*—In a small plant, 2 trucks of 16 cu. ft. capacity are to be pulled up a maximum incline of 1 in 9.

If each truck weighs 650 lb. and its contents 1600 lb., the total load will be 4500 lb. Allowing for friction at  $\frac{1}{80}$  of the load, and for the lifting of the weight up the incline, the working load on the rope will be 590 lb. Multiply this by 5 as the factor of safety, and the product 2950 will be the strength required in the rope in respect of the load upon it. For a rope with 7 wires per strand, Table LXIX. gives the breaking weight for No. 21 gauge at 6809 lb. Then the remainder of this ( $6809 - 2950 = 3859$ ) is available to resist the bending stress, or, in other words, we may allow a bending stress equal to one-third of this, say 1286 lb. Now, turning to Table LXX., it is found that this number lies between those in the column headed 21 gauge, which correspond to a 2 ft. and a 2 ft. 6 in. sheave. It may be concluded, therefore, that a rope of 6 strands of 7/21 could be safely used under the given conditions on a sheave 2 ft. 3 in. in diameter.

1b.—Assuming that we wish to use a still smaller sheave, the next larger size of rope is taken, namely 7/20, which, according to Table LXIX., has a breaking strength of 8619 lb.

Subtracting as before 2950 „

there is an excess of 5669 „, which, divided by 3, gives an allowable bending stress of 1890 lb. This is a little greater than the figure for a 2 ft. sheave, which can therefore be safely used.

Thus, for a given working load and a given small-sized sheave, a rope of large diameter gives better results than a small one, at all events within certain limits. But as a larger rope is heavier to use, it will be profitable to see what is gained by the use of a rope which has 12 wires in each strand.

1c.—As before, select from Table LXIX., but in this case from Column 4, a rope with a breaking load greater than 2950. As a first trial take 12/23 = 6567 lb. Subtracting 2950 gives 3617, which, divided by 3, gives 1206 lb. as the permissible stress due to bending. Turning now to Table LXXI. Column 2, it is found that a 1 ft. 6 in. sheave causes a stress of

1259 lb., and that therefore a sheave slightly larger than this, say 1 ft. 7 in., will be sufficient for this given rope and load.

Collecting the results of these three cases, suitable ropes for the purpose are as follows :

Rope of 42/21 working on a minimum sheave of 2 ft. 3 in.

|         |   |   |         |
|---------|---|---|---------|
| „ 42/20 | „ | „ | 2 „ 0 „ |
| „ 72/23 | „ | „ | 1 „ 7 „ |

*Example 2.*—Suppose a rope to carry a direct pull of 2500 lb. over a minimum sheave of 4 ft. diameter. Assuming first a 42-wire rope.

Direct load  $2500 \times 4 = 10,000$  lb. = breaking strength to resist direct pull.

From Table LXIX. Column 3, select a suitable number, say 20,855, corresponding to 42/17. Then, subtracting 10,000 lb., there is 10,855 lb. to resist bending stress, or an actual allowable stress of 3618 lb. Table LXX. shows in Column 6 that such a rope over a 4-ft. sheave will be subject to a stress of 3500 lb., and is therefore suited to the conditions assumed. Of course, one may not always select the right size from Table LXIX. at the first attempt, but a glance at the table of bending stresses shows whether it is right or wrong, and in any case only two or three trials are necessary.

It is, in fact, far more convenient to use the tables than to construct and use a complicated formula, such as would be necessary to give the required size at once.

It has been pointed out<sup>1</sup> that in cases where the rope is subjected only to a very slight stress, the tension in the rope may be insufficient to bend it to the actual curve of the smallest sheave used, and that in such cases the size of the sheave is of no consequence, as the bending stress depends upon the actual curve of the rope itself. But in the applications of rope haulage now being considered, it may be assumed that the loads put upon the rope in practice will be large enough to bend the rope to the smallest sheave, found by means of the above tables in the manner described. Rollers on which the rope lies slack may be, and are in practice, made smaller than the minimum sheave.

**Rollers.**—The first requirement common to all systems is the provision of a number of horizontal rollers on the track between the rails, to prevent the rope from dragging on the ground or on the sleepers. A convenient form of roller for this purpose is one of cast iron, 4 or 5 in. in diameter and about 11 in. long, with end flanges. It may be supported in hard-wood bearings, bolted down to a sleeper, which may be one of the ordinary track sleepers (see Fig. 195), and must, of course, be fixed and sized so as to be clear of trucks and wheels. Rollers should not be more than 6 yards apart for ropes working under slight tension, as in winch haulages, but for continuous-rope installations they may be from 10 to 15 yards apart. The idle portions of the endless ropes are often carried a few feet above ground on V-pulleys, about 15 in. in diameter, which may be spaced 30 yards or more apart, according to the height available for the ropes to hang between them. At all points

<sup>1</sup> *Engineering Association of South Nashville, Tenn., Proc.*, vol. vii. p. 81.

where there is a convex change of grade a larger roller must be fixed, whose diameter is equal to the minimum calculated by the rules given above.

On roads worked by hauling engines, in passing round curves, a series of similar rollers fixed vertically is used to guide the rope. These should always be of the calculated minimum diameter, because the rope is under tension when pressing against them. This point is often neglected.

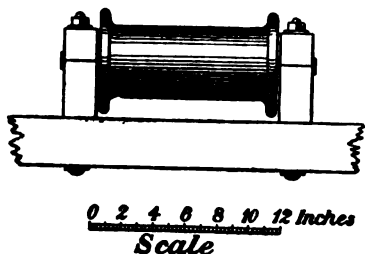


FIG. 195.—Roller.

In endless-rope systems the most important points are, means for giving motion to the rope, for maintaining the tension approximately constant, and for attaching the trucks.

A curved drum, such as shown in Fig. 196, is a simple and convenient device for driving purposes. The curve  $abc$  is so laid out that at one point, say at  $a$ , the surface of the drum is parallel to its axis, while at  $c$  it is nearly at right angles to it, so that it passes through every intermediate angle between  $a$  and  $c$ . The rope is led to the drum at  $b$ , as shown by the arrow

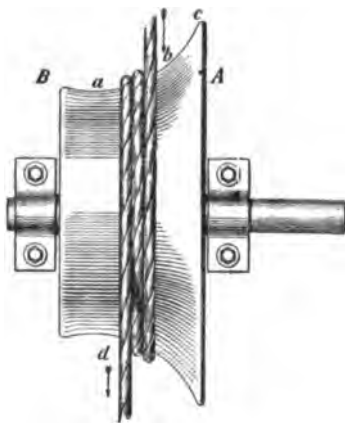


FIG. 196.—Curved Drum.

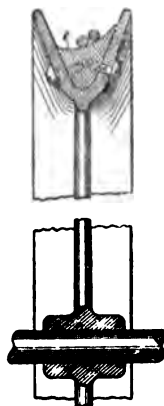


FIG. 197.—Driving Wheel.

0 1 2 3 4 5 6 7 8 9 10 11 12 Inches.

Scale for Figs. 196, 197, 198, 199.

near that end of the drum, which has the largest diameter, and after several complete turns it passes off at  $d$ . It is clear that the continual feeding on at the right-hand side, and paying off at the left-hand, will tend to cause the rope to climb up towards the flange. But this is counteracted by the tendency of the coil as a whole to slip sideways towards  $B$ . Climbing towards  $A$ , however, brings the rope to a steeper part of the drum, and therefore

increases the side slip, while motion towards B brings it to a flatter part, where the side slip is less. Consequently, when either tendency prevails for a time, it automatically shifts the coil on the drum, until the two opposing forces are again balanced. With the drum shown, the rope can be driven in the opposite direction by taking it off and coiling it on in the opposite sense. Or a drum with the curve *abc* repeated on the left-hand side may be used to drive in either direction at will. Wide drums like the one illustrated have been used, but are not necessary when the speed of working is constant, because the actual change of position of the coil is not great. A wheel of the form illustrated in Fig. 197 is therefore suitable. The chief objection to this method of driving is the wear and tear caused to rope and drum by the side slip. With proper lubrication this objection is not serious, and may be lessened as regards the drum by forming its wearing surface of separate

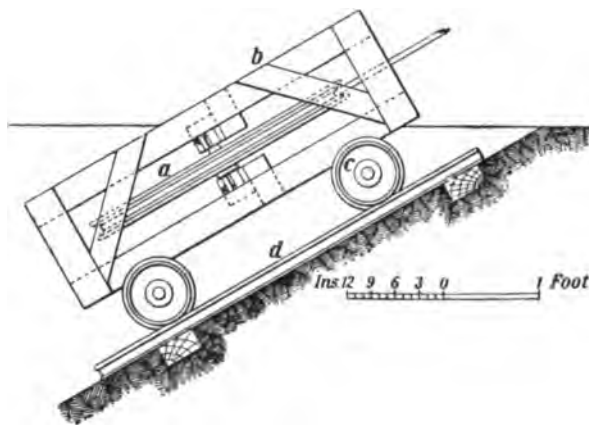


FIG. 198.—Tightening Gear.

replaceable blocks of cast iron, hard-wood, or rubber, as at *e* in Fig. 197. For very elaborate installations, the usual arrangement of tandem V-wheels, as employed for street tramways, may be used with advantage.

The ordinary arrangement of tightening gear is shown in Fig. 198. It consists of a V-wheel *a*, fixed in a frame *b*, supported on wheels *cc*, which run on a steeply inclined track *d*. A second frame loaded with weights may be attached to *b*, or one frame may carry both wheel and weights. This apparatus is perfectly applied to the rope directly it leaves the driving gear.

For overhead ropes, at the points where the trucks come on to the rope, and where they leave it, overhead pulleys are provided, placed sufficiently high to allow a truck and its grip to pass beneath them. The rope hangs, as it were, in a long festoon between these pulleys, from one end of the line to the other, the lowest part resting upon the road rollers. The truck is pushed forward until it reaches a part of the rope low enough to catch in the grip, the latter being properly placed to receive it. At the delivery end of the line, as the rope rises, it lifts out of the grip and leaves the truck free.

Sometimes the same arrangement of overhead rollers is employed at curves to release the truck, which then gravitates round the curve. The rope, after passing round a horizontal sheave into its new direction, and over a second roller, is allowed to descend again and carry the truck forward. This arrangement requires a man to attend to the grips at the point where they pick up the rope again. A similar device is also used for transferring trucks to and from branch lines. An attendant is necessary at each point where the rope leaves the grip, as it sometimes fails to disengage itself soon enough.

Another method of dealing with curves is to place a horizontal V-pulley exactly at the height of the rope when in the grip, and with its outer rim just clear of the grip. As a truck passes, the rope is momentarily drawn out of the V, but it springs back by its own tension as soon as the grip has passed the pulley. Although this device is nominally self-acting, it requires super-

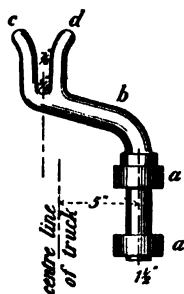


FIG. 199.—Grip.

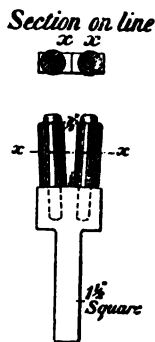


FIG. 200.—Grip.

vision, lest the rope should leave the V-pulley when there is a long interval between two trucks. Occasionally also the rope gets slack in the grip as it passes these pulleys.

The footstep bearing which supports the shaft of each horizontal pulley is sometimes carried on the short arm of a horizontal lever, which has a counterweight on its long arm nearly sufficient to balance the weight of the pulley. A few inches of vertical play is allowed to the pulley to enable it to accommodate itself to slight variations in the height of the grips on the trucks. This arrangement is known as the "Kimberley turn," and is largely used on the Witwatersrand. It also requires attention.

A common form of grip for attaching trucks to rope is shown in Fig. 199, the dimensions there given being suitable for a  $\frac{3}{4}$ -in. rope. The sockets *aa* on the end of the truck should be fixed about 5 in. from the centre of the truck. When the rope falls into the V between *c* and *d*, the grip turns in the socket and jams firmly on the rope, which is thereby slightly bent between the prongs *cd*. This form is generally modified by making the prong *c* shorter than *d* when the trucks have to pass any curves provided with horizontal V-pulleys as described above.

Another device for the same purpose is shown in Fig. 200. This consists of two nearly vertical spindles, slightly inclined towards each other. These are attached to the end of the truck in any suitable manner. Each spindle carries an eccentric sleeve, as shown, which can turn freely upon it. When the rope falls between the sleeves, it turns them round until it is gripped between them. In the figures, the thin sides of the sleeves are turned inwards, and the sizes shown are for a  $\frac{1}{2}$ -in. rope.

Systems of rope haulage, with particular reference to the disengaging of the car on the up or down grade, are shown in Figs. 201 and 202.

**Belt Conveyors.**—The use of this means of transporting material, both in filling and discharging vats, is likely to be considerably extended. It is not

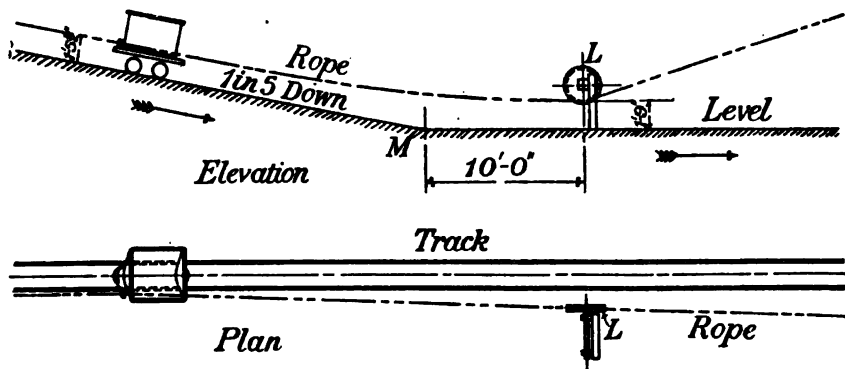


FIG. 201.

necessary to give detailed drawings of the apparatus employed for driving and carrying the belts, because it is advisable to purchase conveyors complete from one of the well-known makers, whose patterns represent the accumulated experience of many years, and of many workers in this special department. The discussion will, therefore, be limited to the general features of the system. Fig. 203 illustrates a method of troughing a conveyor belt with two rollers *b b*, and Fig. 204 shows another arrangement generally employed, in which there is a central horizontal roller *a*, and at each end of it a separate roller *d* on an inclined pin *e*. In either case the returning empty portion of the belt is carried by horizontal rollers *L*, usually three or four, running loose on fixed shafts as in Fig. 204.

The best position for the driving pulley is at the delivery end of the belt, because in that case the loaded part of the belt is under the greater tension, while the return portion runs slack.

The following five recommendations of T. Robins,<sup>1</sup> jun., may be said to represent efficiency in conveyor-belt practice, as required for filling and discharging vats or for dumping filter-press cakes:

1. The driving pulley should not be less than 30 in. in diameter for small

<sup>1</sup> *Trans. Am. Inst. Min. Eng.*, vol. xxvi. p. 78.



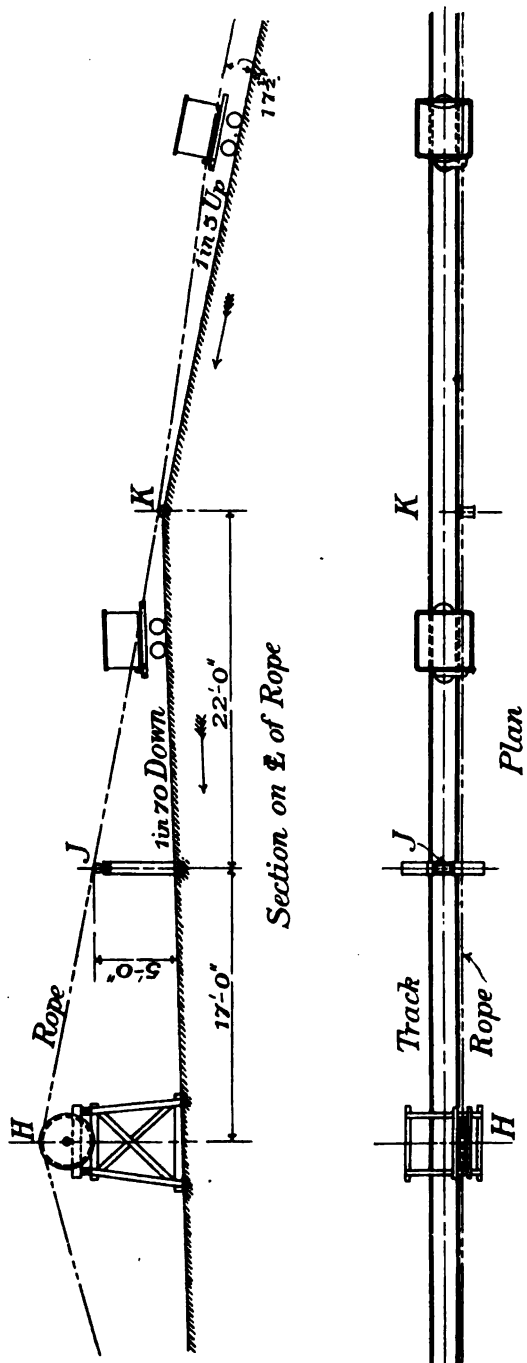


FIG. 202.

belts (say 18 in. wide), and 48 in. for large belts (say 30 in. wide). The pulleys at each end should be 4 in. wider than the belt.

2. The supporting and troughing rollers should be from 4 to 6 ft. apart

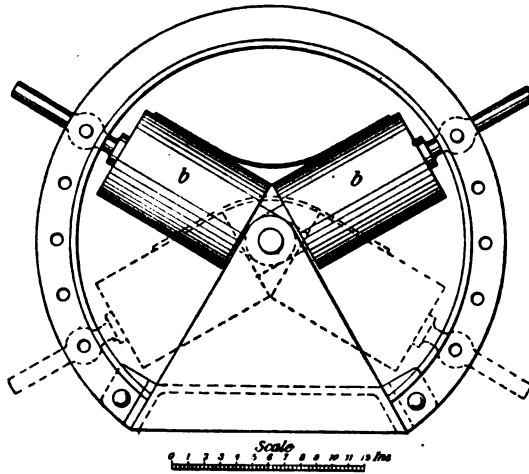


FIG. 203.—Troughing Belt.

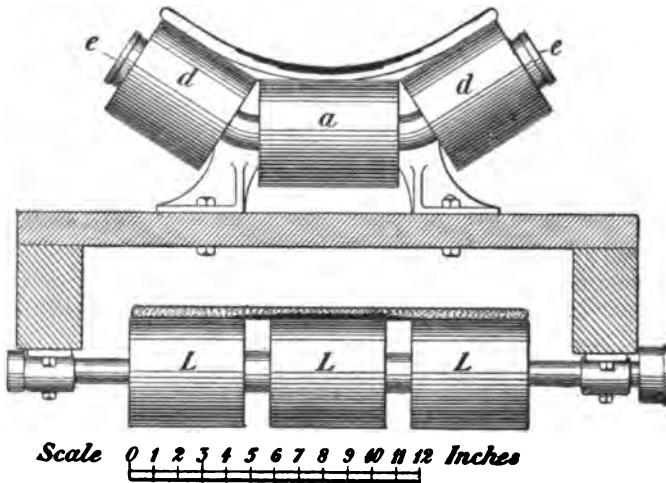


FIG. 204.—Troughing Belt.

between centres, and the rollers under the return portion from 8 to 12 ft. between centres.

3. The thickness of rubber should not be less than  $\frac{1}{4}$  in. over the actual working width of the belt.

*Note.*—The Robins Belt Coy. and other makers supply belts with thicker rubber in the centre than at the sides. This company's method is shown in Fig. 204.

4. The thickness of the canvas should be—

|                    |   |   |   |       |
|--------------------|---|---|---|-------|
| Up to 20 in. wide, | . | . | . | 4 ply |
| „ 24 „             | . | . | . | 5 „   |
| „ 28 „             | . | . | . | 6 „   |
| „ 36 „             | . | . | . | 8 „   |

5. The material should be delivered to the belt by an inclined shoot, and in the direction of the motion of the belt, with as nearly as possible the same speed.

The ordinary form of apparatus for delivering from the belt at any required position is illustrated diagrammatically in Fig. 205.

Two pulleys, *f* and *g*, and a shoot *h*, are fixed in a frame which travels on a track in line with the belt. The loaded part of the belt, which travels in the direction shown by the arrow, passes successively over the pulleys *f* and *g* in the manner shown, so that the material is thrown off into the hopper *i* at the top of the shoot *h*, which latter projects sideways so as to deliver clear of all the gear.

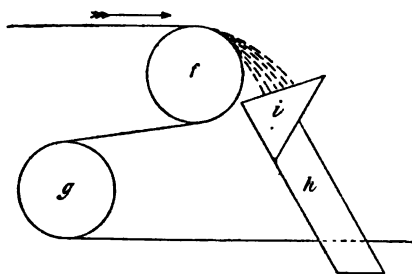


FIG. 205.—Discharging Apparatus.

A shoot may be arranged on either side of the frame, and for large vats may be made adjustable to deliver at various distances from

the belt, and thus lessen the necessity of hand labour for spreading. This frame, called a tripper or discharging carriage, is often made portable and mounted on wheels, so that it may be readily removed from place to place. It is sometimes made to be moved by hand, but is also made so as to be moved by the conveyor belt. It can be arranged to reverse its direction automatically after a short run, thus moving to and fro, delivering, and distributing its load all the time.

Another means for delivering, which is sometimes useful, is shown by the dotted lines in Fig. 204. By moving the rollers into the lower positions there indicated, a convex curvature crosswise is given to the belt, and the material is thereby thrown off on either or both sides.

**Capacity of Belts.**—This depends upon the relative bulk and weight of the material to be conveyed. This point will be made clear by considering that a belt can carry as much coke as can be piled on to it, the only limit being the liability of falling off during transport, but the same belt would break down under an equal *bulk* of shot. Nevertheless, the belt might easily carry a greater *weight* of shot than of coke in a given time. Therefore all general tables of capacity, whether expressed in weight or bulk, are liable to mislead, unless the material to which they apply is clearly stated. Assuming that the strength of the belt is properly proportioned

to its width, the following general formula will apply in all cases by using a suitable coefficient:

Let  $w$  = width of belt in inches.

$v$  = speed in feet per minute.

$K$  = a constant for each individual material, and

$C$  = the capacity in cubic feet per hour.

Then

$$C = Kw^2v.$$

For crushed quartz ore or tailings of ordinary specific gravity, the maximum capacity under perfect conditions as to erection and uniformity of loading is where  $K = 0.016$ . But it is seldom possible to charge the belt with absolute regularity, and it is therefore safer in practice not to expect more than half this amount. The following table is therefore calculated with  $K = 0.008$ . This is a very conservative basis, and the figures given may be somewhat exceeded in cases where ideal conditions of loading obtain, or even when economy in first cost is of exceptional importance.

TABLE LXXIII.—*Approximate Capacity of Troughed Conveyor Belts for Crushed Ore or Tailings, stated in cubic feet delivered per hour.*

| Width<br>in<br>inches. | Speed of Belt in feet per minute. |      |      |      |      |      |      |
|------------------------|-----------------------------------|------|------|------|------|------|------|
|                        | 200                               | 250  | 300  | 350  | 400  | 450  | 500  |
| 16                     | 409                               | 512  | 614  | 717  | 819  | 922  | 1024 |
| 18                     | 528                               | 648  | 778  | 817  | 1057 | 1166 | 1296 |
| 20                     | 640                               | 800  | 960  | 1120 | 1280 | 1440 | 1600 |
| 22                     | 774                               | 968  | 1162 | 1355 | 1549 | 1742 | 1936 |
| 24                     | 921                               | 1152 | 1382 | 1613 | 1843 | 2074 | 2304 |
| 26                     | 1081                              | 1352 | 1622 | 1893 | 2163 | 2434 | 2704 |
| 28                     | 1254                              | 1568 | 1882 | 2195 | 2509 | 2822 | 3136 |
| 30                     | 1440                              | 1800 | 2160 | 2520 | 2880 | 3240 | 3600 |

*Example.*—Suppose a 500-ton vat at 22 cu. ft. per ton has to be filled in 10 hours by a belt running at 350 ft. per minute. Then total cubic feet in vat = 11,000, thus requiring 1100 cu. ft. per hour. The table shows that a 20-in. belt should be provided.

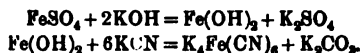
A number of figures relating to the conveyor belts at the New Kleinfontein mine were given by J. W. Forster, the construction engineer. They cover a considerable range of practice, including lengths from 132 to 460 ft., and lifts up to 88 ft., on inclinations ranging between  $0^\circ$  and  $19\frac{1}{2}^\circ$ . From these figures the following empirical formula has been calculated, which will give the h.p. required under actual working conditions. The frictional resistances represented by the first term appear to be practically independent



## APPENDIX.

### ANTIDOTE FOR CYANIDE POISONING.

In 1902 the Victorian (Australia) Government appointed Dr C. J. Martin to investigate the question of antidotes for cyanide poisoning. The result of a number of experiments was to discredit many antidotes that had been considered efficacious but were proved to be worthless, and to suggest the use of freshly prepared ferric hydrate. The original formula was: 1 oz. 23 per cent.  $\text{FeSO}_4$  solution, 1 oz. 5 per cent. KOH solution, and 30 grains of powdered  $\text{MgO}$ . The reactions involved, externally and internally, respectively, are as follows:



The magnesia is used to provide an excess of non-irritant and non-caustic alkali.

Martin's antidote has been universally adopted as the safest and most reliable corrective; and it is advisable, if not customary, to equip a cyanide plant with a number of small, brightly painted emergency cases, placed in conspicuous positions, and containing the essentials for immediate treatment in case of poisoning. Each case should contain:

|                 |                                      |
|-----------------|--------------------------------------|
| 1 stomach tube; | 1 enamelled-iron mug with strainer   |
| 1 mouth gag;    | mouthpiece; and                      |
| 1 spoon;        | Complete set of necessary chemicals. |

The ferrous hydrate decomposes rapidly and must be freshly prepared. It is customary to provide the ingredients for one treatment in the following form:

(A) A hermetically sealed glass tube containing 7-1/2 gm.  $\text{FeSO}_4$  and 30 c.c. water.

(B) A hermetically sealed glass tube containing 1-1/2 gm. NaOH in 30 c.c. water.

(C) A tube containing 2 gm. powdered magnesia.

Immediate action is essential in case of cyanide poisoning. The contents of A and B are broken into the enamelled-iron mug, the magnesia added, and the mixture stirred. The mixture is then drunk, the strainer mouthpiece on the mug preventing any pieces of broken glass from being swallowed; or, if the patient is too far gone, the gag and stomach tube are inserted and the antidote is poured down.

The customary practices of inducing vomiting and cleaning the stomach by means of pump or siphon are equally applicable in the case of cyanide poisoning. The use of hydrogen peroxide is also recommended on account of simplicity, but it has the disadvantage of being slow in action. It may be administered internally (4 oz. or 8 tablespoonsful of 25 per cent.  $\text{H}_2\text{O}_2$ ) or by hypodermic injection (5 per cent. solution). An injection of andrenalin is said to be an efficacious method of delaying the fatal effect of the poison, and gives time for the preparation of an antidote.

TABLE LXXIV.—Showing Contents of Vats in cubic feet.

| Inside<br>Depth in<br>feet and<br>inches. | Inside Diameter of Vat in feet. |       |       |       |       |       |       |       |       |       |       |       |       |       |      |  |  |
|---|---------------------------------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|------|--|--|
|   | 4                               | 5     | 6     | 7     | 8     | 9     | 10    | 11    | 12    | 13    | 14    | 15    | 16    | 17    | 18   |  |  |
| 4 0                                       | 50.3                            | 78.5  | 118.1 | 158.9 | 201.1 | 254.5 | 314.2 | 380.1 | 452.4 | 530.9 | 615.8 | 706.9 | 804.2 | 907.9 | 1018 |  |  |
| 4 3                                       | 53.4                            | 83.4  | 120.2 | 163.6 | 213.6 | 270.4 | 333.8 | 403.9 | 480.7 | 564.1 | 654.2 | 751.0 | 854.5 | 964.7 | 1081 |  |  |
| 4 6                                       | 56.5                            | 88.4  | 127.3 | 173.2 | 226.2 | 286.3 | 353.4 | 427.7 | 508.9 | 597.3 | 692.7 | 795.2 | 904.3 | 1021  | 1145 |  |  |
| 4 9                                       | 59.7                            | 93.3  | 134.3 | 182.8 | 238.3 | 302.2 | 373.1 | 451.4 | 537.2 | 630.5 | 731.3 | 839.4 | 955.0 | 1078  | 1209 |  |  |
| 5 0                                       | 62.8                            | 98.2  | 141.4 | 192.4 | 251.3 | 318.1 | 392.7 | 475.2 | 565.5 | 663.7 | 769.7 | 883.6 | 1005  | 1135  | 1272 |  |  |
| 5 3                                       | 66.0                            | 103.1 | 148.4 | 202.0 | 263.9 | 334.0 | 412.3 | 498.9 | 593.8 | 696.8 | 808.2 | 927.8 | 1058  | 1192  | 1336 |  |  |
| 5 6                                       | 69.1                            | 108.0 | 155.5 | 211.7 | 276.5 | 349.9 | 432.0 | 522.7 | 622.0 | 730.0 | 846.7 | 971.9 | 1106  | 1248  | 1400 |  |  |
| 5 9                                       | 72.3                            | 112.9 | 162.6 | 221.3 | 289.0 | 365.8 | 451.6 | 546.4 | 650.3 | 763.2 | 885.1 | 1016  | 1156  | 1305  | 1463 |  |  |
| 6 0                                       | 75.4                            | 117.8 | 169.3 | 230.9 | 301.6 | 381.7 | 471.2 | 570.2 | 678.6 | 796.4 | 923.6 | 1060  | 1206  | 1362  | 1527 |  |  |
| 6 3                                       | 78.5                            | 122.7 | 176.7 | 240.5 | 314.2 | 397.6 | 490.9 | 594.0 | 706.9 | 829.6 | 962.1 | 1104  | 1257  | 1419  | 1590 |  |  |
| 6 6                                       | 81.7                            | 127.6 | 183.8 | 250.2 | 326.7 | 413.5 | 510.5 | 617.7 | 735.1 | 862.8 | 1001  | 1149  | 1307  | 1475  | 1654 |  |  |
| 6 9                                       | 84.8                            | 132.5 | 190.9 | 259.8 | 339.3 | 429.4 | 530.1 | 641.5 | 763.4 | 895.9 | 1039  | 1193  | 1357  | 1532  | 1718 |  |  |
| 7 0                                       | 88.0                            | 137.4 | 197.9 | 269.4 | 351.9 | 445.3 | 549.8 | 665.2 | 791.7 | 929.1 | 1078  | 1237  | 1407  | 1589  | 1781 |  |  |
| 7 3                                       | 91.1                            | 142.4 | 205.0 | 279.0 | 364.4 | 461.2 | 569.4 | 689.0 | 820.0 | 962.3 | 1116  | 1281  | 1458  | 1646  | 1845 |  |  |
| 7 6                                       | 94.2                            | 147.3 | 212.1 | 288.6 | 377.0 | 477.1 | 589.1 | 712.8 | 848.2 | 995.5 | 1155  | 1325  | 1508  | 1702  | 1909 |  |  |
| 7 9                                       | 97.4                            | 152.2 | 219.1 | 298.3 | 386.6 | 493.0 | 608.7 | 736.5 | 876.5 | 1029  | 1193  | 1370  | 1558  | 1759  | 1972 |  |  |
| 8 0                                       | 100.5                           | 157.1 | 226.2 | 307.9 | 402.1 | 508.9 | 628.3 | 760.3 | 904.3 | 1062  | 1232  | 1414  | 1608  | 1816  | 2036 |  |  |
| 8 3                                       | 103.7                           | 162.0 | 233.3 | 317.5 | 414.7 | 524.8 | 648.0 | 784.0 | 933.1 | 1095  | 1270  | 1458  | 1659  | 1873  | 2099 |  |  |
| 8 6                                       | 106.8                           | 166.9 | 240.3 | 327.1 | 427.3 | 540.7 | 667.6 | 807.8 | 961.3 | 1128  | 1308  | 1502  | 1709  | 1929  | 2163 |  |  |
| 8 9                                       | 110.0                           | 171.8 | 247.4 | 336.7 | 439.3 | 556.7 | 687.2 | 831.5 | 990.0 | 1161  | 1347  | 1546  | 1759  | 1986  | 2227 |  |  |

|      |       |       |       |       |       |       |       |       |      |      |      |      |      |      |      |
|------|-------|-------|-------|-------|-------|-------|-------|-------|------|------|------|------|------|------|------|
| 9 0  | 113.1 | 176.7 | 254.5 | 346.4 | 452.4 | 572.6 | 706.9 | 855.3 | 1018 | 1195 | 1385 | 1590 | 1810 | 2043 | 2290 |
| 9 3  | 116.2 | 181.6 | 261.5 | 356.0 | 465.0 | 588.5 | 726.5 | 879.1 | 1046 | 1228 | 1424 | 1635 | 1860 | 2100 | 2354 |
| 9 6  | 119.4 | 186.5 | 268.6 | 365.6 | 477.5 | 604.4 | 746.1 | 902.8 | 1074 | 1261 | 1462 | 1679 | 1901 | 2156 | 2417 |
| 9 9  | 122.5 | 191.4 | 275.7 | 375.2 | 490.1 | 620.3 | 765.8 | 926.6 | 1108 | 1294 | 1501 | 1723 | 1960 | 2213 | 2481 |
| 10 0 | 125.7 | 196.4 | 282.7 | 384.8 | 502.7 | 636.2 | 785.4 | 950.3 | 1131 | 1327 | 1539 | 1767 | 2011 | 2270 | 2545 |
| 10 3 | 128.8 | 201.3 | 289.8 | 394.5 | 515.2 | 652.1 | 805.0 | 974.1 | 1159 | 1361 | 1578 | 1811 | 2061 | 2327 | 2608 |
| 10 6 | 131.9 | 206.2 | 296.9 | 404.1 | 527.8 | 668.0 | 824.7 | 997.9 | 1188 | 1394 | 1616 | 1856 | 2111 | 2383 | 2672 |
| 10 9 | 135.1 | 211.1 | 303.9 | 413.7 | 540.4 | 683.9 | 844.3 | 1022  | 1216 | 1427 | 1655 | 1900 | 2161 | 2440 | 2736 |
| 11 0 | 138.2 | 216.0 | 311.0 | 423.8 | 552.9 | 699.8 | 863.9 | 1045  | 1244 | 1460 | 1693 | 1944 | 2212 | 2497 | 2799 |
| 11 3 | 141.4 | 220.9 | 318.1 | 433.0 | 565.5 | 715.7 | 883.6 | 1069  | 1272 | 1493 | 1732 | 1986 | 2262 | 2554 | 2863 |
| 11 6 | 144.5 | 225.8 | 325.2 | 442.6 | 578.1 | 731.6 | 903.2 | 1098  | 1301 | 1526 | 1770 | 2032 | 2312 | 2610 | 2926 |
| 11 9 | 147.7 | 230.7 | 332.2 | 452.2 | 590.6 | 747.5 | 922.8 | 1117  | 1329 | 1560 | 1809 | 2076 | 2362 | 2667 | 2990 |
| 12 0 | 150.8 | 235.6 | 339.3 | 461.8 | 603.2 | 763.4 | 942.5 | 1140  | 1357 | 1593 | 1847 | 2121 | 2413 | 2724 | 3054 |
| 12 3 | 153.9 | 240.5 | 346.4 | 471.4 | 615.8 | 779.3 | 962.1 | 1164  | 1385 | 1626 | 1886 | 2165 | 2463 | 2781 | 3117 |
| 12 6 | 157.1 | 245.4 | 353.4 | 481.1 | 628.3 | 795.2 | 981.8 | 1188  | 1414 | 1659 | 1924 | 2209 | 2513 | 2837 | 3181 |
| 12 9 | 160.2 | 250.3 | 360.5 | 490.7 | 640.9 | 811.1 | 1001  | 1212  | 1442 | 1692 | 1963 | 2253 | 2564 | 2894 | 3244 |
| 13 0 | 163.4 | 255.3 | 367.6 | 500.3 | 653.5 | 827.0 | 1021  | 1235  | 1470 | 1726 | 2001 | 2297 | 2614 | 2951 | 3308 |
| 13 6 | 169.6 | 265.1 | 381.7 | 519.5 | 678.6 | 868.8 | 1060  | 1283  | 1527 | 1792 | 2078 | 2386 | 2714 | 3064 | 3435 |
| 14 0 | 175.9 | 274.9 | 395.8 | 538.8 | 703.7 | 890.6 | 1100  | 1330  | 1588 | 1868 | 2166 | 2474 | 2815 | 3178 | 3563 |
| 14 6 | 182.2 | 284.7 | 410.0 | 558.0 | 728.9 | 922.5 | 1139  | 1378  | 1640 | 1925 | 2232 | 2562 | 2915 | 3291 | 3690 |
| 15 0 | 188.5 | 294.5 | 424.1 | 577.3 | 754.0 | 954.3 | 1178  | 1426  | 1696 | 1991 | 2309 | 2651 | 3016 | 3405 | 3817 |
| 15 6 | 194.8 | 303.3 | 438.3 | 596.5 | 779.1 | 986.1 | 1217  | 1473  | 1753 | 2057 | 2386 | 2739 | 3116 | 3518 | 3944 |
| 16 0 | 201.1 | 314.2 | 452.4 | 615.8 | 804.2 | 1018  | 1257  | 1521  | 1810 | 2124 | 2463 | 2827 | 3217 | 3632 | 4072 |



TABLE LXXIV.—continued.

| Inside<br>Depth in<br>feet and<br>inches. | Inside Diameter of Vat in feet. |      |      |      |      |      |      |      |      |      |      |      |      |      |  |
|---|---------------------------------|------|------|------|------|------|------|------|------|------|------|------|------|------|--|
|   | 19                              | 20   | 21   | 22   | 23   | 24   | 25   | 26   | 27   | 28   | 29   | 30   | 31   | 32   |  |
| 4 0                                       | 1134                            | 1257 | 1385 | 1521 | 1662 | 1810 | 1964 | 2124 | 2290 | 2463 | 2642 | 2827 | 3019 | 3217 |  |
| 4 3                                       | 1205                            | 1335 | 1472 | 1616 | 1766 | 1923 | 2086 | 2256 | 2433 | 2617 | 2807 | 3004 | 3208 | 3418 |  |
| 4 6                                       | 1276                            | 1414 | 1559 | 1711 | 1870 | 2036 | 2209 | 2389 | 2577 | 2771 | 2972 | 3181 | 3396 | 3619 |  |
| 4 9                                       | 1347                            | 1492 | 1645 | 1806 | 1974 | 2149 | 2332 | 2522 | 2720 | 2925 | 3137 | 3358 | 3585 | 3820 |  |
| 5 0                                       | 1418                            | 1571 | 1732 | 1901 | 2077 | 2262 | 2454 | 2655 | 2863 | 3079 | 3303 | 3534 | 3774 | 4021 |  |
| 5 3                                       | 1489                            | 1649 | 1818 | 1996 | 2181 | 2375 | 2577 | 2787 | 3006 | 3233 | 3468 | 3711 | 3963 | 4222 |  |
| 5 6                                       | 1559                            | 1728 | 1905 | 2091 | 2285 | 2488 | 2700 | 2920 | 3149 | 3387 | 3633 | 3888 | 4151 | 4423 |  |
| 5 9                                       | 1630                            | 1806 | 1992 | 2186 | 2389 | 2601 | 2823 | 3053 | 3292 | 3541 | 3798 | 4064 | 4340 | 4624 |  |
| 6 0                                       | 1701                            | 1885 | 2078 | 2281 | 2493 | 2714 | 2945 | 3186 | 3435 | 3695 | 3963 | 4241 | 4529 | 4825 |  |
| 6 3                                       | 1772                            | 1964 | 2165 | 2376 | 2597 | 2827 | 3068 | 3318 | 3578 | 3848 | 4128 | 4418 | 4717 | 5027 |  |
| 6 6                                       | 1843                            | 2042 | 2251 | 2471 | 2701 | 2941 | 3191 | 3451 | 3722 | 4002 | 4293 | 4595 | 4906 | 5228 |  |
| 6 9                                       | 1914                            | 2121 | 2338 | 2566 | 2804 | 3054 | 3313 | 3584 | 3865 | 4156 | 4459 | 4771 | 5095 | 5429 |  |
| 7 0                                       | 1985                            | 2199 | 2425 | 2661 | 2908 | 3167 | 3436 | 3717 | 4008 | 4310 | 4624 | 4948 | 5283 | 5630 |  |
| 7 3                                       | 2056                            | 2278 | 2511 | 2756 | 3012 | 3280 | 3559 | 3849 | 4151 | 4464 | 4789 | 5125 | 5472 | 5831 |  |
| 7 6                                       | 2126                            | 2356 | 2598 | 2851 | 3116 | 3393 | 3682 | 3982 | 4294 | 4618 | 4954 | 5301 | 5661 | 6032 |  |
| 7 9                                       | 2197                            | 2435 | 2684 | 2946 | 3220 | 3506 | 3804 | 4115 | 4437 | 4772 | 5119 | 5478 | 5849 | 6238 |  |
| 8 0                                       | 2268                            | 2513 | 2771 | 3041 | 3324 | 3619 | 3927 | 4247 | 4580 | 4926 | 5284 | 5655 | 6033 | 6434 |  |
| 8 3                                       | 2339                            | 2592 | 2857 | 3136 | 3428 | 3732 | 4050 | 4380 | 4724 | 5080 | 5449 | 5832 | 6227 | 6635 |  |
| 8 6                                       | 2410                            | 2670 | 2944 | 3231 | 3532 | 3845 | 4172 | 4513 | 4867 | 5234 | 5614 | 6008 | 6416 | 6836 |  |
| 8 9                                       | 2481                            | 2749 | 3031 | 3326 | 3635 | 3958 | 4295 | 4646 | 5010 | 5388 | 5780 | 6185 | 6604 | 7037 |  |

|      |      |      |      |      |      |      |      |      |      |      |       |       |       |       |
|------|------|------|------|------|------|------|------|------|------|------|-------|-------|-------|-------|
| 9 0  | 2552 | 2827 | 3117 | 3421 | 3789 | 4072 | 4418 | 4778 | 5138 | 5542 | 5945  | 6362  | 6798  | 7238  |
| 9 3  | 2623 | 2906 | 3204 | 3516 | 3848 | 4185 | 4541 | 4911 | 5296 | 5696 | 6110  | 6538  | 6982  | 7439  |
| 9 6  | 2694 | 2985 | 3290 | 3611 | 3947 | 4298 | 4668 | 5044 | 5439 | 5850 | 6275  | 6715  | 7170  | 7640  |
| 9 9  | 2764 | 3068 | 3377 | 3706 | 4051 | 4411 | 4786 | 5177 | 5582 | 6004 | 6440  | 6892  | 7359  | 7841  |
| 10 0 | 2835 | 3142 | 3464 | 3801 | 4155 | 4524 | 4909 | 5309 | 5726 | 6168 | 6605  | 7069  | 7548  | 8042  |
| 10 3 | 2906 | 3220 | 3550 | 3896 | 4259 | 4637 | 5031 | 5442 | 5869 | 6311 | 6770  | 7245  | 7736  | 8244  |
| 10 6 | 2977 | 3299 | 3637 | 3991 | 4362 | 4750 | 5164 | 5575 | 6012 | 6465 | 6935  | 7422  | 7925  | 8445  |
| 10 9 | 3048 | 3377 | 3728 | 4086 | 4466 | 4868 | 5277 | 5707 | 6156 | 6619 | 7101  | 7599  | 8114  | 8646  |
| 11 0 | 3119 | 3456 | 3810 | 4181 | 4570 | 4976 | 5400 | 5840 | 6298 | 6773 | 7266  | 7775  | 8302  | 8847  |
| 11 3 | 3190 | 3534 | 3897 | 4276 | 4674 | 5089 | 5522 | 5973 | 6441 | 6927 | 7431  | 7952  | 8491  | 9048  |
| 11 6 | 3261 | 3618 | 3988 | 4372 | 4778 | 5202 | 5645 | 6106 | 6584 | 7081 | 7596  | 8129  | 8680  | 9249  |
| 11 9 | 3331 | 3691 | 4070 | 4467 | 4882 | 5316 | 5768 | 6238 | 6728 | 7235 | 7761  | 8306  | 8869  | 9450  |
| 12 0 | 3402 | 3770 | 4156 | 4562 | 4986 | 5429 | 5891 | 6371 | 6871 | 7389 | 7926  | 8482  | 9057  | 9651  |
| 12 3 | 3473 | 3848 | 4243 | 4657 | 5090 | 5542 | 6013 | 6504 | 7014 | 7543 | 8091  | 8659  | 9246  | 9852  |
| 12 6 | 3544 | 3927 | 4330 | 4752 | 5193 | 5655 | 6136 | 6637 | 7157 | 7697 | 8257  | 8836  | 9435  | 10053 |
| 12 9 | 3615 | 4006 | 4416 | 4847 | 5297 | 5768 | 6259 | 6769 | 7300 | 7851 | 8422  | 9012  | 9628  | 10264 |
| 13 0 | 3686 | 4084 | 4508 | 4942 | 5401 | 5881 | 6381 | 6902 | 7443 | 8005 | 8587  | 9189  | 9812  | 10455 |
| 13 6 | 3828 | 4241 | 4676 | 5132 | 5609 | 6107 | 6627 | 7168 | 7730 | 8313 | 8917  | 9543  | 10189 | 10857 |
| 14 0 | 3969 | 4398 | 4849 | 5323 | 5817 | 6333 | 6872 | 7433 | 8016 | 8621 | 9247  | 9896  | 10567 | 11269 |
| 14 6 | 4111 | 4565 | 5022 | 5512 | 6024 | 6560 | 7118 | 7698 | 8302 | 8928 | 9578  | 10249 | 10944 | 11662 |
| 15 0 | 4253 | 4712 | 5195 | 5702 | 6232 | 6786 | 7363 | 7964 | 8598 | 9236 | 9908  | 10608 | 11322 | 12064 |
| 15 6 | 4395 | 4869 | 5369 | 5892 | 6440 | 7012 | 7609 | 8229 | 8875 | 9544 | 10238 | 10956 | 11699 | 12466 |
| 16 0 | 4536 | 5027 | 5542 | 6082 | 6646 | 7238 | 7854 | 8495 | 9161 | 9852 | 10568 | 11310 | 12076 | 12868 |

TABLE LXXIV.—*continued.*

| Inside Diameter of Vat in feet.           |      |      |      |      |      |      |       |       |       |       |       |       |       |       |
|---|------|------|------|------|------|------|-------|-------|-------|-------|-------|-------|-------|-------|
| Inside<br>Depth in<br>feet and<br>inches. | 33   | 34   | 35   | 36   | 37   | 38   | 39    | 40    | 41    | 42    | 43    | 44    | 45    | 50    |
| ' 4 0                                     | 3421 | 3632 | 3848 | 4071 | 4301 | 4536 | 4775  | 5027  | 5281  | 5542  | 5809  | 6082  | 6362  | 7854  |
| 4 3                                       | 3635 | 3859 | 4089 | 4326 | 4570 | 4820 | 5077  | 5341  | 5611  | 5888  | 6172  | 6462  | 6759  | 8345  |
| 4 6                                       | 3849 | 4086 | 4330 | 4580 | 4838 | 5104 | 5376  | 5655  | 5941  | 6234  | 6535  | 6842  | 7157  | 8836  |
| 4 9                                       | 4063 | 4313 | 4570 | 4835 | 5107 | 5387 | 5674  | 5969  | 6271  | 6581  | 6898  | 7223  | 7555  | 9327  |
| 5 0                                       | 4277 | 4540 | 4811 | 5089 | 5376 | 5671 | 5973  | 6283  | 6601  | 6927  | 7261  | 7603  | 7952  | 9818  |
| 5 3                                       | 4490 | 4767 | 5051 | 5344 | 5645 | 5954 | 6272  | 6597  | 6931  | 7274  | 7624  | 7983  | 8350  | 10308 |
| 5 6                                       | 4704 | 4994 | 5292 | 5598 | 5914 | 6238 | 6570  | 6912  | 7261  | 7620  | 7987  | 8363  | 8747  | 10799 |
| 5 9                                       | 4918 | 5221 | 5532 | 5853 | 6182 | 6521 | 6869  | 7226  | 7591  | 7966  | 8350  | 8743  | 9145  | 11290 |
| 6 0                                       | 5132 | 5448 | 5773 | 6107 | 6451 | 6805 | 7168  | 7540  | 7922  | 8313  | 8713  | 9123  | 9543  | 11781 |
| 6 3                                       | 5346 | 5675 | 6013 | 6362 | 6720 | 7088 | 7466  | 7854  | 8252  | 8659  | 9076  | 9503  | 9940  | 12272 |
| 6 6                                       | 5559 | 5901 | 6254 | 6616 | 6989 | 7372 | 7765  | 8168  | 8582  | 9005  | 9439  | 9883  | 10338 | 12763 |
| 6 9                                       | 5773 | 6128 | 6494 | 6871 | 7258 | 7655 | 8063  | 8482  | 8912  | 9352  | 9802  | 10264 | 10735 | 13254 |
| 7 0                                       | 5987 | 6355 | 6735 | 7125 | 7526 | 7939 | 8362  | 8796  | 9242  | 9698  | 10165 | 10644 | 11133 | 13745 |
| 7 3                                       | 6201 | 6582 | 6975 | 7380 | 7795 | 8222 | 8661  | 9111  | 9572  | 10044 | 10528 | 11024 | 11531 | 14235 |
| 7 6                                       | 6415 | 6809 | 7216 | 7634 | 8064 | 8506 | 8959  | 9425  | 9902  | 10391 | 10892 | 11404 | 11928 | 14726 |
| 7 9                                       | 6629 | 7036 | 7456 | 7888 | 8333 | 8789 | 9258  | 9739  | 10232 | 10737 | 11255 | 11784 | 12326 | 15217 |
| 8 0                                       | 6842 | 7263 | 7697 | 8143 | 8602 | 9073 | 9557  | 10053 | 10562 | 11084 | 11618 | 12164 | 12723 | 15708 |
| 8 3                                       | 7056 | 7490 | 7937 | 8397 | 8870 | 9356 | 9855  | 10367 | 10892 | 11430 | 11981 | 12544 | 13121 | 16199 |
| 8 6                                       | 7270 | 7717 | 8178 | 8652 | 9139 | 9640 | 10154 | 10681 | 11222 | 11776 | 12344 | 12925 | 13519 | 16600 |
| 8 9                                       | 7484 | 7944 | 8419 | 8906 | 9408 | 9924 | 10453 | 10996 | 11552 | 12123 | 12707 | 13305 | 13916 | 17181 |

|      |       |       |       |       |       |       |       |       |       |       |       |       |       |       |
|------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|-------|
| 9 0  | 7698  | 8171  | 8659  | 9161  | 9677  | 10207 | 10751 | 11310 | 11882 | 12469 | 13070 | 13685 | 14314 | 17672 |
| 9 3  | 7912  | 8398  | 8900  | 9415  | 9946  | 10491 | 11050 | 11624 | 12212 | 12815 | 13438 | 14065 | 14712 | 18162 |
| 9 6  | 8126  | 8625  | 9140  | 9670  | 10214 | 10774 | 11349 | 11938 | 12542 | 13162 | 13796 | 14445 | 15109 | 18653 |
| 9 9  | 8339  | 8852  | 9381  | 9924  | 10483 | 11058 | 11647 | 12252 | 12872 | 13508 | 14159 | 14825 | 15507 | 19144 |
| 10 0 | 8553  | 9079  | 9621  | 10179 | 10753 | 11341 | 11946 | 12566 | 13202 | 13854 | 14522 | 15205 | 15904 | 19635 |
| 10 3 | 8767  | 9306  | 9862  | 10433 | 11021 | 11625 | 12245 | 12881 | 13533 | 14201 | 14885 | 15585 | 16302 | 20126 |
| 10 6 | 8981  | 9533  | 10102 | 10688 | 11290 | 11908 | 12543 | 13195 | 13863 | 14547 | 15248 | 15966 | 16700 | 20617 |
| 10 9 | 9194  | 9760  | 10343 | 10942 | 11559 | 12192 | 12842 | 13509 | 14193 | 14893 | 15611 | 16346 | 17067 | 21108 |
| 11 0 | 9408  | 9987  | 10583 | 11197 | 11827 | 12475 | 13140 | 13823 | 14523 | 15240 | 15974 | 16726 | 17495 | 21599 |
| 11 3 | 9622  | 10214 | 10824 | 11451 | 12096 | 12759 | 13439 | 14137 | 14853 | 15586 | 16337 | 17106 | 17892 | 22089 |
| 11 6 | 9836  | 10441 | 11064 | 11706 | 12365 | 13042 | 13738 | 14451 | 15183 | 15933 | 16700 | 17486 | 18290 | 22580 |
| 11 9 | 10050 | 10668 | 11305 | 11960 | 12634 | 13326 | 14036 | 14766 | 15513 | 16279 | 17063 | 17866 | 18688 | 23071 |
| 12 0 | 10264 | 10895 | 11545 | 12214 | 12903 | 13609 | 14355 | 15080 | 15843 | 16625 | 17426 | 18246 | 19086 | 23562 |
| 12 3 | 10477 | 11122 | 11786 | 12469 | 13171 | 13893 | 14634 | 15394 | 16173 | 16972 | 17790 | 18627 | 19483 | 24063 |
| 12 6 | 10691 | 11349 | 12026 | 12723 | 13440 | 14176 | 14932 | 15708 | 16503 | 17318 | 18153 | 19007 | 19880 | 24544 |
| 12 9 | 10905 | 11576 | 12267 | 12978 | 13709 | 14460 | 15231 | 16022 | 16833 | 17664 | 18516 | 19387 | 20278 | 25035 |
| 13 0 | 11119 | 11803 | 12507 | 13232 | 13978 | 14744 | 15530 | 16336 | 17163 | 18011 | 18879 | 19767 | 20676 | 25526 |
| 13 6 | 11547 | 12257 | 12989 | 13741 | 14515 | 15311 | 16127 | 16965 | 17823 | 18703 | 19605 | 20527 | 21471 | 26507 |
| 14 0 | 11974 | 12711 | 13470 | 14250 | 15053 | 15878 | 16724 | 17593 | 18483 | 19396 | 20331 | 21287 | 22266 | 27489 |
| 14 6 | 12402 | 13165 | 13951 | 14759 | 15591 | 16445 | 17322 | 18221 | 19144 | 20089 | 21057 | 22048 | 23061 | 28471 |
| 15 0 | 12830 | 13619 | 14432 | 15268 | 16128 | 17012 | 17919 | 18850 | 19804 | 20782 | 21783 | 22803 | 23857 | 29453 |
| 15 6 | 13267 | 14073 | 14913 | 15777 | 16666 | 17579 | 18516 | 19478 | 20464 | 21474 | 22509 | 23568 | 24652 | 30434 |
| 16 0 | 13685 | 14527 | 15394 | 16286 | 17203 | 18146 | 19113 | 20106 | 21124 | 22167 | 23235 | 24323 | 25447 | 31416 |

TABLE LXXV.—*Discharge of Battery Pulp from short lengths of pipe with small heads. Useful in designing distributors and in connection with hose-filling.*

| Head<br>in feet. | Length<br>in feet. | Discharge in cubic feet per minute. |      |                |                |       |                |       |       | Length<br>in feet. | Head<br>in feet. |     |
|------------------|--------------------|-------------------------------------|------|----------------|----------------|-------|----------------|-------|-------|--------------------|------------------|-----|
|                  |                    | Diameter of Pipe in inches.         |      |                |                |       |                |       |       |                    |                  |     |
|                  |                    | $\frac{3}{4}$                       | 1    | $1\frac{1}{4}$ | $1\frac{1}{2}$ | 2     | $2\frac{1}{2}$ | 3     | 4     |                    |                  |     |
| 5                | 1                  | 1.622                               | 3.83 | 5.82           | 9.18           | 18.84 | ...            | ...   | ..    | 1                  | 5                |     |
|                  | 2                  | 1.147                               | 2.86 | 4.11           | 6.49           | 13.32 | ...            | ...   | ...   | 2                  |                  |     |
|                  | 4                  | .811                                | 1.67 | 2.91           | 4.59           | 9.42  | ...            | ...   | ...   | 4                  |                  |     |
|                  | 6                  | .662                                | 1.36 | 2.38           | 3.75           | 7.69  | ...            | ...   | ...   | 6                  |                  |     |
|                  | 8                  | .574                                | 1.18 | 2.06           | 3.24           | 6.66  | ...            | ...   | ...   | 8                  |                  |     |
|                  | 10                 | .513                                | 1.05 | 1.84           | 2.90           | 5.96  | ...            | ...   | ...   | 10                 |                  |     |
|                  | 15                 | .419                                | .86  | 1.50           | 2.37           | 4.86  | ...            | ...   | ...   | 15                 |                  |     |
|                  | 20                 | .363                                | .74  | 1.30           | 2.05           | 4.21  | ...            | ...   | ...   | 20                 |                  |     |
| 7.5              | 2                  | 1.405                               | 2.88 | 5.04           | 7.95           | 16.32 | ...            | ...   | ...   | 2                  | 7.5              |     |
|                  | 4                  | .994                                | 2.04 | 3.56           | 5.62           | 11.54 | ...            | ...   | ...   | 4                  |                  |     |
|                  | 6                  | .811                                | 1.67 | 2.91           | 4.59           | 9.42  | ...            | ...   | ...   | 6                  |                  |     |
|                  | 8                  | .703                                | 1.44 | 2.52           | 3.97           | 8.16  | ...            | ...   | ...   | 8                  |                  |     |
|                  | 10                 | .628                                | 1.29 | 2.25           | 3.55           | 7.30  | 12.75          | ...   | ...   | 10                 |                  |     |
|                  | 15                 | .513                                | 1.05 | 1.84           | 2.90           | 5.96  | 10.41          | ...   | ...   | 15                 |                  |     |
|                  | 20                 | .444                                | .91  | 1.59           | 2.51           | 5.16  | 9.01           | ...   | ...   | 20                 |                  |     |
|                  | 2                  | 1.622                               | 3.83 | 5.82           | 9.18           | 18.84 | 32.91          | ...   | ...   | 2                  |                  | 1   |
| 4                | 1.147              | 2.86                                | 4.11 | 6.49           | 13.32          | 23.27 | ...            | ...   | 4     |                    |                  |     |
| 6                | .937               | 1.92                                | 3.36 | 5.30           | 10.88          | 19.00 | ...            | ...   | 6     |                    |                  |     |
| 8                | .811               | 1.67                                | 2.91 | 4.59           | 9.42           | 16.46 | ...            | ...   | 8     |                    |                  |     |
| 10               | .726               | 1.49                                | 2.60 | 4.10           | 8.43           | 14.72 | 23.22          | 47.66 | 10    |                    |                  |     |
| 15               | .592               | 1.22                                | 2.12 | 3.35           | 6.88           | 12.02 | 18.96          | 38.92 | 15    |                    |                  |     |
| 20               | .513               | 1.05                                | 1.84 | 2.90           | 5.96           | 10.41 | 16.42          | 33.70 | 20    |                    |                  |     |
| 25               | ...                | .94                                 | 1.65 | 2.60           | 5.33           | 9.31  | 14.68          | 30.14 | 25    |                    |                  |     |
| 1.5              | 30                 | ...                                 | .86  | 1.50           | 2.37           | 4.86  | 8.50           | 13.40 | 27.52 | 30                 | 1.5              |     |
|                  | 40                 | ...                                 | .74  | 1.30           | 2.05           | 4.21  | 7.36           | 11.61 | 23.88 | 40                 |                  |     |
|                  | 10                 | ...                                 | 1.82 | 3.19           | 5.03           | 10.32 | 18.03          | 28.44 | 58.37 | 10                 |                  |     |
|                  | 15                 | ...                                 | 1.49 | 2.60           | 4.10           | 8.43  | 14.72          | 23.22 | 47.66 | 15                 |                  |     |
|                  | 20                 | ...                                 | 1.29 | 2.25           | 3.55           | 7.30  | 12.74          | 20.11 | 41.28 | 20                 |                  |     |
|                  | 25                 | ...                                 | 1.15 | 2.02           | 3.18           | 6.53  | 11.40          | 17.98 | 36.92 | 25                 |                  |     |
|                  | 30                 | ...                                 | 1.05 | 1.84           | 2.90           | 5.96  | 10.41          | 16.42 | 33.70 | 30                 |                  |     |
|                  | 40                 | ...                                 | .91  | 1.59           | 2.51           | 5.16  | 9.01           | 14.22 | 29.19 | 40                 |                  |     |
| 2                | 10                 | ...                                 | 2.11 | 3.68           | 5.80           | 11.92 | 20.82          | 32.84 | 67.40 | 10                 | 2                |     |
|                  | 15                 | ...                                 | 1.72 | 3.00           | 4.74           | 9.73  | 17.00          | 26.81 | 55.04 | 15                 |                  |     |
|                  | 20                 | ...                                 | 1.49 | 2.60           | 4.10           | 8.43  | 14.72          | 23.22 | 47.66 | 20                 |                  |     |
|                  | 25                 | ...                                 | 1.33 | 2.33           | 3.67           | 7.54  | 13.16          | 20.77 | 42.63 | 25                 |                  |     |
|                  | 30                 | ...                                 | 1.22 | 2.12           | 3.35           | 6.88  | 12.02          | 18.96 | 38.92 | 30                 |                  |     |
|                  | 40                 | ...                                 | 1.05 | 1.84           | 2.90           | 5.96  | 10.41          | 16.42 | 33.70 | 40                 |                  |     |
|                  | 10                 | ...                                 | 2.36 | 4.11           | 6.49           | 13.32 | 23.27          | 36.71 | 75.36 | 10                 |                  | 2.5 |
|                  | 15                 | ...                                 | 1.92 | 3.36           | 5.30           | 10.88 | 19.00          | 29.97 | 61.53 | 15                 |                  |     |
| 20               | ...                | 1.66                                | 2.91 | 4.59           | 9.42           | 16.46 | 25.96          | 53.29 | 20    |                    |                  |     |
| 25               | ...                | 1.49                                | 2.60 | 4.10           | 8.43           | 14.72 | 23.22          | 47.66 | 25    |                    |                  |     |
| 30               | ...                | 1.36                                | 2.38 | 3.75           | 7.69           | 13.44 | 21.20          | 43.51 | 30    |                    |                  |     |
| 40               | ...                | 1.18                                | 2.06 | 3.24           | 6.66           | 11.63 | 18.36          | 37.68 | 40    |                    |                  |     |
| 50               | ...                | 1.05                                | 1.84 | 2.90           | 5.96           | 10.41 | 16.42          | 33.70 | 50    |                    |                  |     |
| 3                | 10                 | ...                                 | 2.68 | 4.51           | 7.11           | 14.59 | 25.49          | 40.21 | 82.55 | 10                 | 3                |     |
|                  | 15                 | ...                                 | 2.11 | 3.68           | 5.80           | 11.92 | 20.82          | 32.84 | 67.40 | 15                 |                  |     |
|                  | 20                 | ...                                 | 1.82 | 3.19           | 5.03           | 10.32 | 18.03          | 28.44 | 58.37 | 20                 |                  |     |
|                  | 25                 | ...                                 | 1.63 | 2.85           | 4.50           | 9.23  | 16.12          | 25.43 | 52.21 | 25                 |                  |     |
|                  | 30                 | ...                                 | 1.49 | 2.60           | 4.10           | 8.43  | 14.72          | 23.22 | 47.66 | 30                 |                  |     |
|                  | 40                 | ...                                 | 1.29 | 2.25           | 3.55           | 7.30  | 12.74          | 20.11 | 41.28 | 40                 |                  |     |
|                  | 50                 | ...                                 | 1.15 | 2.02           | 3.18           | 6.53  | 11.40          | 17.98 | 36.92 | 50                 |                  |     |



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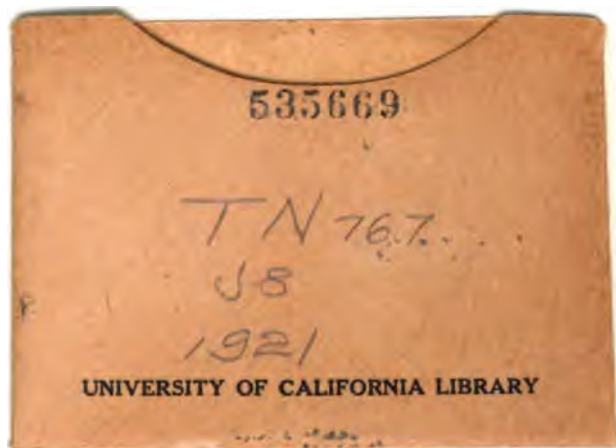
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